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POCKET-BOOK FOR
MINERS AND METALLURGISTS

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COMPILED BY

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FIFTH EDITION

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PREFACE TO FIFTH EDITION

THE original plan of this book, published fifty-eight years ago, although slightly modified, has been generally adhered to. The whole of the text of this edition, however, has been re-written by the original author to accord with modern thought and ideas. It is therefore entirely up to date, and it is hoped will prove as useful and as popular as it was when it was first brought out.

F. D. P.

PREFACE TO FIRST EDITION

THIS little work is intended as a vade mecum for those engaged in mining and metallurgical industries. It is designed to furnish them with a collection of rules, formulae, tables, and other such information as may be found useful, if only to refresh the memory, in the multifarious operations falling within the daily duties of the practical miner and metallurgist.

In field work, especially, persons connected with mining have long felt the want of a suitable POCKET BOOK which should contain concise notes of the various sciences with which metal-mining is concerned—a volume which they could carry about with them on the field, when travelling, or when engaged away from their headquarters and thus debarred from consulting works of reference. The notes and other information comprised in the following pages have been put together in their present form in the spare time of the compiler. Having been found of use to him in the actual exercise of his profession, they are now published in the hope that they may be found of like service to others.

Such a work as this is of necessity a compilation. A list of books is accordingly given to which reference has been made. Where the author of a table or formula previously published is known, care has been taken to give due acknowledgment. But it is not always easy to discover the original authority for material current in various publications.

The compiler's thanks are due to Messrs. S. J. Becher and W. Harris for their assistance in checking certain calculations contained in the work, and to the former gentleman, also, for his "Hints on Photography." He is also indebted to Mr. E. Henry Davies for having revised the proof sheets.

The compiler will feel indebted to anyone who will kindly assist him by pointing out any errors which may possibly have escaped his observation, in order that they may be corrected in future editions. He will also welcome any suggestions for the improvement of the work.

F. D. P.

MELBOURNE (VICTORIA)
March, 1892

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ARITHMETIC

Multiplication

Arrange the work so that the figures of the multiplier are used from left to right. In the case of decimals it is unnecessary to write the figures to the right of the line as shown. Drop the right-hand figure of the multiplicand one by one as the multiplication of each line is completed by successive figures of the multiplier, making proper allowance while doing so in the product.

Example. Multiply 45·32 by 3·618.

$$\begin{array}{r|l}
\multicolumn{1}{r}{4532} & \\
\multicolumn{1}{r}{3618} & \\
\hline
13596 & \\
2719 & 2 \\
45 & 32 \\
36 & 256 \\
\hline
163{\cdot}96 & 776 \\
\hline\hline
\end{array}$$

Division

Work may be abbreviated by cutting off one figure of the divisor at each division, instead of adding a zero to the dividend.

Example. Divide 3618 into 4532.

$$\begin{array}{rrl}
3618 & 4532 & 1 \\
 & 3618 & \\
\cline{2-2}
361 & 914 & 2 \\
 & 722 & \\
\cline{2-2}
36 & 192 & 5 \\
 & 180 & \\
\cline{2-2}
3 & 12 & 4 \\
 & 12 & \\
\cline{2-2}
 & .. & \\
\end{array}$$

Fractions

Addition of Vulgar Fractions

$$\frac{1}{2}+\frac{1}{2}=\frac{2}{2}=1 \quad \frac{6}{8}+\frac{3}{8}=\frac{9}{8}=1\tfrac{1}{8} \quad \frac{3}{7}+\frac{2}{3}=\frac{9+14}{21}=\frac{23}{21}=1\tfrac{2}{21}$$

Subtraction of Vulgar Fractions

$$\frac{1}{2} - \frac{1}{2} = 0 \qquad \frac{6}{8} - \frac{2}{8} = \frac{4}{8} = \frac{1}{2}$$

$$3\tfrac{4}{7} - 1\tfrac{2}{3} = \frac{25}{7} - \frac{5}{3} = \frac{75 - 35}{21} = \frac{40}{21} = 1\tfrac{19}{21}$$

Multiplication of Vulgar Fractions

$$\frac{1}{2} \times \frac{1}{2} = \frac{1}{4} \qquad \frac{4}{5} \times \frac{2}{3} = \frac{8}{15} \qquad \frac{2}{3} \times \frac{3}{7} \times \frac{1}{2} = \frac{6}{42} = \frac{1}{7}$$

Division of Vulgar Fractions

$$\frac{1}{2} \div \frac{1}{2} = \frac{2}{2} = 1 \qquad \frac{4}{5} \div \frac{2}{3} = \frac{12}{10} = \frac{6}{5} = 1\tfrac{1}{5}$$

To find the greatest common divisor of a Vulgar Fraction

```
Thus 70/175          70)175(2
                        140
                        ---
                         35)70(2            Ans. 0·35
                            70
                            --
                            ..
```

To reduce a Vulgar Fraction to its lowest terms. Find the greatest common divisor and divide both the numerator and denominator by it.

Thus $\frac{70}{175}$, whose greatest common divisor is 35 = $\frac{2}{5}$.

To reduce a Vulgar Fraction to a decimal form. Divide the numerator by the denominator.

```
Thus 8/25            25)8·0(0·32
                          75
                          --
                           50              Ans. 0·32
                           50
                           --
                           ..
```

Example. Reduce 3 inches to the decimal of 1 foot, i.e. 12 inches. The question is therefore reduce $\frac{3}{12}$ to a decimal as above.

Example. How many feet are there in 0·75 of a yard?

```
         0·75
            3 feet in a yard
         ----
   Feet  2·25
           12 inches in a foot
         ----
    In.  3·00
                              Ans. 2 ft 3 in.
```

Decimals

Add together 0·27 0·439 0·073 0·01

$$\begin{array}{l} 0{\cdot}27 \\ 0{\cdot}439 \\ 0{\cdot}073 \\ 0{\cdot}01 \\ \hline 0{\cdot}792 \text{ Ans.} \end{array}$$

Subtract 0·23 from 0·94; also 0·0001 from 1·3.

$$\begin{array}{l} 0{\cdot}94 \\ 0{\cdot}23 \\ \hline 0{\cdot}71 \end{array} \qquad \begin{array}{l} 1{\cdot}3 \\ 0{\cdot}0001 \\ \hline 1{\cdot}2999 \end{array}$$

Multiply 0·734 by 1·02.

$$\begin{array}{r} 0{\cdot}734 \\ 1{\cdot}02 \\ \hline 1468 \\ 734 \\ \hline 0{\cdot}74868 \end{array}$$

Divide 14·357 by 0·07.

$$\begin{array}{l} 7)1435{\cdot}7(205{\cdot}1 \\ \underline{14} \\ 35 \\ 35 \\ 7 \\ \underline{7} \\ \cdot\cdot \end{array}$$

Evolution or Extraction of Roots

Square root expressed thus, $\sqrt{36} = 6$.

Rule. Separate the given number into periods of two figures each, by placing a point over the units figure, then over every second towards the left in whole numbers, and over every second figure towards the right in decimals and proceed thus—

$\sqrt{5329}$;

$$\begin{array}{r} 5\dot{3}2\dot{9}(73 \\ 49 \\ 143)\;429 \\ 429 \\ \cdot\cdot\cdot \end{array}$$

Ans. 73

$\sqrt{7\cdot3441}$;

$$\begin{array}{rr}
 & \dot{7}\cdot3\dot{4}4\dot{1}(2\cdot71 \\
 & 4 \\ \cline{2-2}
47) & 334 \\
 & 329 \\ \cline{2-2}
541) & 541 \\
 & 541 \\ \cline{2-2}
 & \ldots
\end{array}$$

Ans. 2·71

Note. For square and cube roots see Tables. Fourth, fifth roots, etc., are best done by logarithms.

ALGEBRA

Powers, Roots

$$a^m . a^n = a^{m+n};\ (a . b)^n = a^n . b^n;\ a^m \div a^n = a^{m-n};\ a^{-n} = \frac{1}{a^n};$$

$$(a^n)^m = a^{n \cdot m};\ \left(\frac{a}{b}\right)^m = \frac{a^m}{b^m};\ a^{-m} . a^n = a^{-m+n};\ (a^m)^{-n} = a^{-m \cdot n};$$

$$\frac{a^2 - b^2}{a - b} = a + b;\ \frac{a^2 - b^2}{a + b} = a - b;\ (a + b)(a - b) = a^2 - b^2;$$

$$a^0 = 1;\ (\sqrt[n]{a})^n = \sqrt[n]{a^n} = a;\ \sqrt[n]{a . b} = \sqrt[n]{a} . \sqrt[n]{b};$$

$$\sqrt[n]{\frac{a}{b}} = \frac{\sqrt[n]{a}}{\sqrt[n]{b}};\ \sqrt[n]{a^m} = a^{\frac{m}{n}};\ \sqrt[n]{a^m} = \sqrt[n.n]{a^{m \cdot n}};\ \sqrt[n]{\sqrt[m]{a}} = \sqrt[n.m]{a}.$$

Equations

1st degree with one unknown quantity

$$ax \times b = 0 \ldots x = -\frac{b}{a};$$

1st degree with two unknown quantities

$$\left\{\begin{array}{l} ax + by = c \\ a_1x + b_1y = c_1 \end{array}\right\} \ \ldots \ \left\{\begin{array}{l} x = \dfrac{b_1c - bc_1}{ab_1 - a_1b}; \\ y = \dfrac{ac_1 - a_1c}{ab_1 - a_1b}; \end{array}\right.$$

2nd degree, common form,

$$ax^2 + bx + c = 0 \quad x = \frac{-b \pm \sqrt{(b^2 - 4ac)}}{2a}$$

2nd degree, normal form,

$$x^2 + px + q = 0 \quad x = -\frac{p}{2} \pm \sqrt{\left(\frac{p^2}{4} - q\right)};$$

3rd degree,

$$Ax^3 + Bx^2 + Cx + D = 0 \quad x^3 + ax^2 + bx + c = 0$$

Arithmetical Progression. Let a be the first term, l the last term, d the common difference, n the number of terms, and s the sum of n terms. Then—

$$l = a + (n - 1)d \qquad a = 1 - (n - 1)d \qquad n = \frac{1 - a}{d} + 1$$

$$s = \frac{n}{2}(a + 1) \qquad d = \frac{1 - a}{n - 1}$$

Geometrical Progression. Let a be the first term, l the last term, r the common ratio, n the number of terms, and s the sum of n terms. Then—

$$l = ar^{n-1}$$

$$s = \frac{a(r^n - 1)}{r - 1}$$

If n is infinity and r^2 less than unity—

$$s = \frac{a}{1 - r}$$

Permutations. If M denotes the number of permutations of n things taken p at a time—

$$M = n(n - 1)(n - 2) \ . \ . \ . \ (n - p + 1)$$

Combinations. If M denotes the number of combinations of n things taken p at a time—

$$M = \frac{n(n - 1)(n - 2) \ . \ . \ . \ (n - p + 1)}{p}$$

TRIGONOMETRY

Trigonometric Functions of a Right-angled Triangle

A, B, and C are the vertices. C the right-angle.

$$\sin A = \frac{\text{opposite side}}{\text{hypotenuse}} = \frac{a}{c}$$

$$\tan A = \frac{\text{opposite side}}{\text{adjacent side}} = \frac{a}{b}$$

$$\cos A = \frac{\text{adjacent side}}{\text{hypotenuse}} = \frac{b}{c}$$

$$\operatorname{cosec} A = \frac{\text{hypotenuse}}{\text{opposite side}} = \frac{c}{a}$$

$$\cot A = \frac{\text{adjacent side}}{\text{opposite side}} = \frac{b}{a}$$

$$\sec A = \frac{\text{hypotenuse}}{\text{adjacent side}} = \frac{c}{b}$$

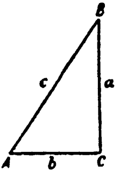


FIG. 1. RIGHT-ANGLED TRIANGLE

Between	0°	and	90°	the sin is	+	cos	+	tan	+	cot	+
,,	90°	,,	180°	,,	+	,,	−	,,	−	,,	−
,,	180°	,,	270°	,,	−	,,	−	,,	+	,,	+
,,	270°	,,	360°	,,	−	,,	+	,,	−	,,	−

Formulae

$$\sin\alpha = \sin(180-\alpha) = -\sin(180+\alpha) = -\sin(360-\alpha) = \cos(90-\alpha)\left\{\begin{matrix}\geqq -1\\ \leqq +1\end{matrix}\right\}$$

$$\cos\alpha = -\cos(180-\alpha) = -\cos(180+\alpha) = +\cos(360-\alpha) = \sin(90-\alpha)\left\{\begin{matrix}\geqq -1\\ \leqq +1\end{matrix}\right\}$$

$\sin(-\alpha) = -\sin\alpha$;
$\cos(-\alpha) = +\cos\alpha$;

$$\tan\alpha = \frac{\sin\alpha}{\cos\alpha}\left\{\begin{matrix}-\infty\\ \text{to}\\ +\infty\end{matrix}\right\}; \qquad \cot\alpha = \frac{\cos\alpha}{\sin\alpha}\left\{\begin{matrix}-\infty\\ \text{to}\\ +\infty\end{matrix}\right\};$$

$\tan\alpha \,.\, \cot\alpha = 1$;
$\sin^2\alpha + \cos^2\alpha = 1$; $\sin\alpha = \sqrt{(1-\cos^2\alpha)}$; $\cos\alpha = \sqrt{(1-\sin^2\alpha)}$;
$\sin 2\alpha = 2\sin\alpha\cos\alpha$; $\cos 2\alpha = \cos^2\alpha - \sin^2\alpha$;
$\sin(\alpha \pm \beta) = \sin\alpha\cos\beta \pm \cos\alpha\sin\beta$; $\sec\alpha.\,\cos\alpha = 1$;
$\cos(\alpha \pm \beta) = \cos\alpha\cos\beta \mp \sin\alpha\sin\beta$; $\mathrm{cosec}\,\alpha.\,\sin\alpha = 1$;

$$\tan(\alpha \pm \beta) = \frac{\tan\alpha \pm \tan\beta}{1 \pm \tan\alpha . \tan\beta} \quad \cot(\alpha \pm \beta) = \frac{\cot\alpha\cot\beta \mp 1}{\cot\alpha \pm \cot\beta};$$

$$\tan 2\alpha = \frac{2\tan\alpha}{1 - \tan^2\alpha}; \quad \sin \tfrac{1}{2}a = \pm\sqrt{\frac{(1 - \cos\alpha)}{2}};$$

$$\cos \tfrac{1}{2}\alpha = \pm\sqrt{\frac{(1 + \cos\alpha)}{2}}:$$

$$\sin 2\alpha + \sin 2\beta = 2\sin(\alpha + \beta)\cos(\alpha - \beta);$$

$$\sin 2\alpha - \sin 2\beta = 2\sin(\alpha - \beta)\cos(\alpha + \beta);$$

$$\cos 2\alpha + \cos 2\beta = 2\cos(\alpha + \beta)\cos(\alpha - \beta);$$

$$\cos 2\alpha - \cos 2\beta = -2\sin(\alpha + \beta)\sin(\alpha - \beta);$$

$$\sin\alpha = \frac{\tan\alpha}{\sqrt{(1 + \tan^2\alpha)}} = \frac{1}{\sqrt{(1 + \cot^2\alpha)}} = \frac{1}{\operatorname{cosec}\alpha};$$

$$\cos\alpha = \frac{1}{\sqrt{(1 + \tan^2\alpha)}} = \frac{\cot\alpha}{\sqrt{(1 + \cot^2\alpha)}} = \frac{1}{\sec\alpha}$$

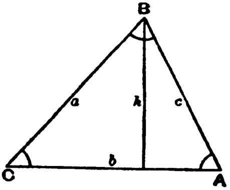


FIG. 2. OBLIQUE-ANGLED TRIANGLE

In the oblique angled triangle, A, B, C—

(1) $\dfrac{a}{\sin A} = \dfrac{b}{\sin B} = \dfrac{c}{\sin C}$

(2) $a^2 = b^2 + c^2 - 2bc \cos A.$

(3) $\tan\dfrac{A}{2} = \sqrt{\left\{\dfrac{(a + b - c)(a + c - b)}{(a + b + c)(b + c - a)}\right\}}$

(4) $\dfrac{a + b}{a - b} = \dfrac{\tan\dfrac{A + B}{2}}{\tan\dfrac{A - B}{2}}$

(5) Area of the triangle, A, B, C $= \frac{bb}{2} = \frac{ab \sin C}{2} = \frac{c^2 \sin B \sin A}{2 \sin C} = \sqrt{\{s(s-a)(s-b)(s-c)\}}$

where h = height

s = sum of the sides divided by 2.

EXAMPLES

Given	Required
1. One side and two adjacent angles a, B, C	b and c by equation (1) $A = 180 - (B + C)$ Area from equation (5)
2. One side, one adjacent and one opposite angle a, B, A	b and c by equation (1) $C = 180 - (A + B)$ Area from equation (5)
3. Three sides a, b, c	Two angles by (2) or (3) When the third will be 180 − the other two Area by (5)
4. Two sides and the included angle b, c, A	$B = \frac{B+C}{2} + \frac{B-C}{2}$ $C = \frac{B+C}{2} - \frac{B-C}{2}$ $\frac{B+C}{2} = \frac{180-A}{2}$; $\frac{B-C}{2}$ from (4) A from equation (1) Area from equation (5)
5. Two sides and the angle opposite the greater side b, c, B	**By equation (4)**

MENSURATION

Measurement of Mathematical Figures

s = side; b = base; h = height; d = diameter; r = radius; c = circumference; a = area; V = volume; $\pi = 3{\cdot}1416$; e = length of arc; o = chord; α = central angle.

Triangles. $a = \frac{1}{2}b \times h$.

Squares. $a = s^2$.

Parallelograms. $a = b \times h$.

Trapezoids. $a = \dfrac{\text{the two parallel } s \times h}{2}$.

Trapeziums. a = divide into triangles and add their areas together.

Polygons. $a = s \times h$ (from centre of figure to centre of side) × half the number of sides.

Circles. $a = \pi r^2 = \pi \dfrac{d^2}{4}$; $c = 2\pi r = \pi d$; $d = \dfrac{c}{\pi} = \dfrac{c \times 7}{22}$

To find the diameter of a circle equal in area to a given square, multiply one side of the square by 1·12838.

To find the side of a square equal in area to a given circle, multiply the diameter by 0·88623.

Sectors. $a = \dfrac{e \times r}{2} = \dfrac{\alpha}{360} \pi r^2 = \dfrac{\alpha}{360} \pi \dfrac{d^2}{4}$;

$$e = \frac{\alpha}{360} \pi 2r = \frac{\alpha}{360} \pi d.$$

Segments. $a = \left(\dfrac{\alpha}{360} 2\pi - \sin \alpha\right) \dfrac{r^2}{2}$

For flat segments a = about $\frac{2}{3}oh$; $e = o\left(1 + \dfrac{h}{3r}\right)$

Circular rings. a = area of the smaller circle subtracted from the area of the greater.

Ellipse. a = product of diameters × 0·7854.

Parabola. $a = \frac{2}{3}b \times h$.

Cubes. $V = s^3$.

Prisms. V = area of base × h.

Pyramids. $V = \dfrac{\text{area of base} \times h}{3}$

Prismoids. Add together the areas of the two parallel surfaces, and four times the area of the section taken halfway between them and parallel to them; multiply the sum by the perpendicular distance between the two parallel sides, and divide the product by 6.

Wedges. $V = \frac{(\text{length of edge} + 2s \text{ parallel to the edge})\, h \times s \text{ at right angles to edge}}{6}$

Cylinders. $V = \pi r^2 h.$

Cones. $V = \frac{\text{area of base} \times h}{3}$

To find the surface of any regular cone, multiply the circumference of its base by the *slant* height; take half the product.

Sphere. $V = 0{\cdot}5236d^3 = 4{\cdot}1888r^3 = \frac{4}{3}\pi r^3 = \frac{1}{6}\pi d^3$. Surface $= \pi d^2 = 3{\cdot}1416d^2 = 4\pi r^2 = 12{\cdot}5664r^2$.

Spherical segment. Volume = square of the radius of its base multiplied by 3, add to the product the square of its height, multiply the sum by the height, and multiply this last product by 0·5236.

Cylindrical rings. Volume = area of cross section of bar of which ring is made × ½ sum of inner and outer diameters × 3·141593. Surface = circumference of bar of which ring is made × ½ sum of inner and outer diameters × 3·141593.

Paraboloid. $V = \frac{1}{2}\pi r^2 h.$

LOGARITHMS

The base of common or Briggs's logarithms is ten, as being convenient for the decimal system. Each log is supposed to have a decimal sign before it. The integer or characteristic is the whole number on the left of the decimal point. The decimal portion, which is always positive, is called the mantissa. The characteristic of a logarithm of any number greater than unity is one less than the number of digits in the integral part of the number. The characteristic of the logarithm of any number less than unity is negatively and numerically one greater than the number of ciphers before the first significant figure of the number when expressed as a decimal. The negative is designated by a bar placed over the characteristic.

Example.

The log of	1946	is	3·28914	The log of	0·1946	is	$\bar{1}$·28914
,, ,,	194·6	,,	2·28914	,, ,,	0·0195	,,	$\bar{2}$·28914
,, ,,	19·46	,,	1·28914	,, ,,	0·0019	,,	$\bar{3}$·28914
,, ,,	1·946	,,	0·28914	,, ,,	0·0002	,,	$\bar{4}$·28914

To add two negative characteristics, take their sum and make it negative. Thus, $\bar{2}$ added to $\bar{3}$ makes $\bar{5}$.

To add a positive and negative characteristic, take their difference and call it positive or negative, whichever is the greater. Thus 6 and $\bar{2}$ give 4; while $\bar{5}$ and 2 give $\bar{3}$.

To subtract a negative characteristic, change its sign to plus, then add it. Thus subtract $\bar{3}$ from 2 gives 5; while $\bar{5}$ subtracted from $\bar{2}$ gives 3.

The common logarithm of all numbers expressed by the same figures in the same order with the decimal point in different positions have the same mantissa, but the characteristics are different.

Example.

The log of	50	is	1·69897	The log of	5·0	is	$\bar{1}$·69897
,, ,,	500	,,	2·69897	,, ,,	0·5	,,	$\bar{2}$·69897
,, ,,	5000	,,	3·69897	,, ,,	0·05	,,	$\bar{3}$·69897

For most practical purposes four figure logarithms are sufficient. To find the logarithm of any number containing more than three figures, place a decimal point after the second figure from the left, thus changing the mantissa into an integer and a decimal. If the decimal portion contains more than two figures and the second figure is 5 or more than 5, add 1 to the first figure of the decimal. Find the difference between the nearest logarithm and the next highest; this difference is the tenth part of the unit. Multiply the difference by the figure whose mantissa is sought, and add it to the decimal of the mantissa already found.

Example. What is the logarithm of 1946?

The log of 194 is	28780
The log of 195 is	29003
The difference is	223

223 × 6 = 1338. Add 134 to 28780 and we get 28914, the log required.

To multiply a logarithm with a negative characteristic, multiply the mantissa in the ordinary way, then multiply the negative characteristic, which will give a negative product. If there is any carriage over from the decimal portion, add it to the characteristic; then according to the rule, take their difference and consider the result to be positive or negative, whichever is the greater.

EXAMPLE. Multiply $\bar{3}\cdot 60638$ by 4. Here we have $3 \times 4 = 12$, with a carriage of 2 gives $\overline{10}\cdot 81914$.

To divide a logarithm having a negative characteristic. If the characteristic is divisible by the divisor, write down the quotient with a negative sign, and divide the fractional part by common rule. But if the negative characteristic is not divisible by the divisor, add such a negative number to it as will make it divisible, and prefix an equal positive integer to the fractional part of the logarithm; then divide the increased negative exponent and the other part of the logarithm separately, and the former quotient, taken negatively, will be the characteristic to the fractional part of the quotient. Thus divide $\bar{9}$ by 3, gives $\bar{3}$. Divide $\overline{10}$ by 3; one must add $\bar{2}$ to the $\overline{10}$ making it $\overline{12}$ into which 3 will go $\bar{4}$ times; the quotient is therefore $\bar{4}\cdot 2/3$.

To multiply by logarithms, add them together, and find the corresponding number. Log $ab = \log a + \log b$.

To divide by logarithms, subtract one from the other and find the corresponding number. Log $\frac{a}{b} = \log a - \log b$.

To raise a number to any power, multiply the log by the index of the power, and find the corresponding number. Log $a^n = n \times \log a$.

To extract the root, divide the log by the index of the root, and find the corresponding number. Log $\frac{\sqrt[n]{a}}{n} = \frac{\log a}{n}$

To find the natural number corresponding to any logarithm, look to the column headed *Nr.* for the first two figures of the mantissa; the third figure is to be found in one of the vertical columns headed 0 to 9. If it is exactly found, the number must be made to correspond to the characteristic by pointing off decimals or annexing ciphers. If the decimal portion cannot be found exactly, find the next lower logarithm, subtract it from the given logarithm, divide the difference from the difference between the next lower and the next

LOGARITHMS

Nr.	0	1	2	3	4	5	6	7	8	9	Diff.
10	00000	00432	00860	01284	01703	02119	02531	02938	03342	03743	396
11	04139	04532	04922	05308	05690	06070	06446	06819	07188	07555	363
12	07918	08279	08636	08991	09342	09691	10037	10380	10721	11059	335
13	11394	11727	12057	12385	12710	13033	13354	13672	13988	14301	312
14	14613	14922	15229	15534	15836	16137	16435	16732	17026	17319	290
15	17609	17898	18184	18469	18752	19033	19312	19590	19866	20140	272
16	20412	20683	20952	21219	21484	21748	22011	22272	22531	22789	256
17	23045	23300	23553	23805	24055	24304	24551	24797	25042	25285	242
18	25527	25768	26007	26245	26482	26717	26951	27184	27416	27646	229
19	27875	28103	28330	28556	28780	29003	29226	29447	29667	29885	218
20	30103	30320	30535	30750	30963	31175	31387	31597	31806	32015	207
21	32222	32428	32634	32838	33041	33244	33445	33646	33846	34044	198
22	34242	34439	34635	34830	35025	35218	35411	35603	35793	35984	189
23	36173	36361	36549	36736	36922	37107	37291	37475	37658	37840	181
24	38021	38202	38382	38561	38739	38917	39094	39270	39445	39620	174
25	39794	39967	40140	40312	40483	40654	40824	40993	41162	41330	167
26	41497	41664	41830	41996	42160	42325	42488	42651	42813	42975	161
27	43136	43297	43457	43616	43775	43933	44091	44248	44404	44560	156
28	44716	44871	45025	45179	45332	45484	45637	45788	45939	46090	150
29	46240	46389	46538	46687	46835	46982	47129	47276	47422	47567	145
30	47712	47857	48001	48144	48287	48430	48572	48714	48855	48996	140
31	49136	49276	49415	49554	49693	49831	49969	50106	50243	50379	136
32	50515	50651	50786	50920	51055	51188	51322	51455	51587	51720	132
33	51851	51083	52114	52244	52375	52504	52634	52763	52892	53020	128
34	53148	53275	53403	53529	53656	53782	53908	54033	54158	54283	124
35	54407	54531	54654	54777	54900	55023	55145	55267	55388	55509	121
36	55630	55751	55871	55991	56110	56229	56348	56467	56585	56703	117
37	56820	56937	57054	57171	57287	57403	57519	57634	57749	57864	114
38	57978	58092	58206	58320	58433	58546	58659	58771	58883	58995	111
39	59106	59218	59329	59439	59550	59660	59770	59879	59988	60097	109
40	60206	60314	60423	60531	60638	60746	60853	60959	61066	61172	106
41	61278	61384	61490	61595	61700	61805	61909	62014	62118	62221	104
42	62325	62428	62531	62634	62737	62839	62941	63043	63144	63246	101
43	63347	63448	63548	63649	63749	63849	63949	64048	64147	64246	99
44	64345	64444	64542	64640	64738	64836	64933	65031	65128	65225	97
45	65321	65418	65514	65610	65706	65801	65896	65992	66087	66181	95
46	66276	66370	66464	66558	66652	66745	66839	66932	67025	67117	93
47	67210	67302	67394	67486	67578	67669	67761	67852	67943	68034	90
48	68124	68215	68305	68395	68485	68574	68664	68753	68842	68931	89
49	69020	69108	69197	69285	69373	69461	69548	69636	69723	69810	87
50	69897	69984	70070	70157	70243	70329	70415	70501	70586	70672	86
51	70757	70842	70927	71012	71096	71181	71265	71349	71433	71517	84
52	71600	71684	71767	71850	71933	72016	72099	72181	72263	72346	83
53	72428	72509	72591	72673	72754	72835	72916	72997	73078	73159	81
54	73239	73320	73400	73480	73560	73640	73719	73799	73878	73957	80
55	74036	74115	74194	74273	74351	74429	74507	74586	74663	74741	78
56	74819	74896	74974	75051	75128	75205	75282	75358	75435	75511	77
57	75587	75664	75740	75815	75891	75967	76042	76118	76193	76268	76
58	76343	76418	76492	76567	76641	76716	76790	76864	76938	77012	74
59	77085	77159	77232	77305	77379	77452	77525	77597	77670	77743	73
60	77815	77887	77960	78032	78104	78176	78247	78319	78390	78462	72
61	78533	78604	78675	78746	78817	78888	78958	79029	79099	79169	71
62	79239	79309	79379	79449	79518	79588	79657	79727	79796	79865	69
63	79934	80003	80072	80140	80209	80277	80346	80414	80482	80550	68
64	80618	80686	80754	80821	80889	80956	81023	81090	81158	81224	67
Nr.	0	1	2	3	4	5	6	7	8	9	Diff.

LOGARITHMS (*continued*)

Nr.	0	1	2	3	4	5	6	7	8	9	Diff.
65	81291	81358	81425	81491	81558	81624	81690	81757	81823	81889	**66**
66	81954	82020	82086	82151	82217	82282	82347	82413	82478	82543	**65**
67	82607	82672	82737	82802	82866	82930	82995	83059	83123	83187	**64**
68	83251	83315	83378	83442	83506	83569	83632	83696	83759	83822	**63**
69	83885	83948	84011	84073	84136	84198	84261	84323	84386	84448	**63**
70	84510	84572	84634	84696	84757	84819	84880	84942	85003	85065	**62**
71	85126	85187	85248	85309	85370	85431	85491	85552	85612	85673	**61**
72	85733	85794	85854	85914	85974	86034	86094	86153	86213	86273	**60**
73	86332	86392	86451	86510	86570	86629	86688	86747	86806	86864	**59**
74	86923	86982	87040	87099	87157	87216	87274	87332	87390	87448	**58**
75	87506	87564	87622	87679	87737	87795	87852	87910	87967	88024	**58**
76	88081	88138	88195	88252	88309	88366	88423	88480	88536	88593	**57**
77	88649	88705	88762	88818	88874	88930	88986	89042	89098	89154	**56**
78	89209	89265	89321	89376	89432	89487	89542	89597	89653	89708	**55**
79	89763	89818	89873	89927	89982	90037	90091	90146	90200	90255	**55**
80	90309	90363	90417	90472	90526	90580	90634	90687	90741	90795	**54**
81	90849	90902	90956	91009	91062	91116	91169	91222	91275	91328	**53**
82	91381	91434	91487	91540	91593	91645	91698	91751	91803	91855	**52**
83	91908	91960	92012	92065	92117	92169	92221	92273	92324	92376	**52**
84	92428	92480	92531	92583	92634	92686	92737	92788	92840	92891	**51**
85	92942	92993	93044	93095	93146	93197	93247	93298	93349	93399	**51**
86	93450	93500	93551	93601	93651	93702	93752	93802	93852	93902	**50**
87	93952	94002	94052	94101	94151	94201	94250	94300	94349	94399	**50**
88	94448	94498	94547	94596	94645	94694	94743	94792	94841	94890	**49**
89	94939	94988	95036	95085	95134	95182	95231	95279	95328	95376	**49**
90	95424	95472	95521	95569	95617	95665	95713	95761	95809	95856	**48**
91	95904	95952	95999	96047	96095	96142	96190	96237	96284	96332	**47**
92	96379	96426	96473	96520	96567	96614	96661	96708	96755	96802	**47**
93	96848	96895	96942	96988	97035	97081	97128	97174	97220	97267	**46**
94	97313	97359	97405	97451	97497	97543	97589	97635	97681	97727	**46**
95	97772	97818	97864	97909	97955	98000	98046	98091	98137	98182	**45**
96	98227	98272	98318	98363	98408	98453	98498	98543	98588	98632	**45**
97	98677	98722	98767	98811	98856	98900	98945	98989	99034	99078	**45**
98	99123	99167	99211	99255	99300	99344	99388	99432	99476	99520	**44**
99	99564	99607	99651	99695	99739	99782	99826	99870	99913	99957	**44**
100	00000	00043	00087	00130	00173	00217	00260	00303	00346	00389	**43**
101	00432	00475	00518	00561	00604	00647	00689	00732	00775	00817	**43**
102	00860	00903	00945	00988	01030	01072	01115	01157	01199	01242	**42**
103	01284	01326	01368	01410	01452	01494	01536	01578	01620	01662	**42**
104	01703	01745	01787	01828	01870	01912	01953	01995	02036	02078	**42**
105	02119	02160	02202	02243	02284	02325	02366	02407	02449	02490	**41**
106	02531	02572	02612	02653	02694	02735	02776	02816	02857	02898	**41**
107	02938	02979	03019	03060	03100	03141	03181	03222	03262	03302	**41**
108	03342	03383	03423	03463	03503	03543	03583	03623	03663	03703	**40**
109	03743	03782	03822	03862	03902	03941	03981	04021	04060	04100	**40**
110	04139	04179	04218	04258	04297	04336	04376	04415	04454	04493	**39**
111	04532	04571	04610	04650	04689	04727	04766	04805	04844	04883	**39**
112	04922	04961	04999	05038	05077	05115	05154	05192	05231	05269	**39**
113	05308	05346	05385	05428	05461	05500	05538	05576	05614	05652	**38**
114	05690	05729	05767	05805	05843	05881	05918	05956	05094	06032	**38**
115	06070	06108	06145	06183	06221	06258	06296	06333	06371	06408	**38**
116	06446	06483	06521	06558	06595	06638	06670	06707	06744	06781	**37**
117	06819	06856	06893	06930	06967	07004	07041	07078	07115	07151	**37**
118	07188	07225	07262	07298	07335	07372	07408	07445	07482	07518	**37**
119	07555	07591	07628	07664	07700	07737	07773	07809	07846	07882	**36**
Nr.	**0**	**1**	**2**	**3**	**4**	**5**	**6**	**7**	**8**	**9**	**Diff.**

higher logarithms and annex the quotient to the natural number found opposite the lower logarithm.

Example. What is the number corresponding to the log 3·28914?

Next higher log	29003	Given log	28914
Next lower log	28780	Next lower log	28780
Difference	223	Difference	134

$134 \div 223 = 6$, log $28780 = 194$, annex 6 to that and we get 1946.

Logarithms of Trigonometrical Functions

To find the logarithms of sines, cosines, tangents and cotangents from 0° to 45° look for degrees in the left-hand column, and the minutes on the top line. To find the sine, cosine, tangent and cotangent between 45° and 90° read upwards from the bottom, the degrees being in the right-hand column, and note that the order of the minutes on the bottom line is the reverse of that on the top. If the angle is between 90° and 180° take the angle from 180°; if between 180° and 270° take 180° from the angle; and if between 270° and 360° take 270° from 360° and the remainder from the table. The tables only give the figures for every 10 minutes; should intermediate minutes be sought, subtract the next lower from the next higher figures in the columns, multiply the odd minutes by the difference. For sines and tangents, add the product to the nearest lower figure found. In the case of cosines and cotangents, subtract the difference.

Examples. Sin 25° 17′. Cosine 25° 17′.

25° 10′	0·425	25° 10′	0·9050
$428 - 425 = 3$		$905 - 904 = 1$	
$3 \times 7 =$	21	$1 \times 7 =$	7
	0·4271		0·9043

TABLE OF NATURAL SINES AND TANGENTS TO RADIUS 1

Deg.	Sine						
	0′	10′	20′	30′	40′	50′	
0	0·000	0·003	0·006	0·009	0·012	0·015	**89**
1	0·017	0·020	0·023	0·026	0·029	0·032	**88**
2	0·035	0·038	0·041	0·044	0·047	0·049	**87**
3	0·052	0·055	0·058	0·061	0·064	0·067	**86**
4	0·070	0·073	0·076	0·078	0·081	0·084	**85**
5	0·087	0·090	0·093	0·096	0·099	0·102	**84**
6	0·105	0·107	0·110	0·113	0·116	0·119	**83**
7	0·122	0·125	0·128	0·131	0·133	0·136	**82**
8	0·139	0·142	0·145	0·148	0·151	0·154	**81**
9	0·156	0·159	0·162	0·165	0·168	0·171	**80**
10	0·174	0·177	0·179	0·182	0·185	0·188	**79**
11	0·191	0·194	0·197	0·199	0·202	0·205	**78**
12	0·208	0·211	0·214	0·216	0·219	0·222	**77**
13	0·225	0·228	0·231	0·233	0·236	0·239	**76**
14	0·242	0·245	0·248	0·250	0·253	0·256	**75**
15	0·259	0·262	0·264	0·267	0·270	0·273	**74**
16	0·276	0·278	0·281	0·284	0·287	0·290	**73**
17	0·292	0·295	0·298	0·301	0·303	0·306	**72**
18	0·309	0·312	0·315	0·317	0·320	0·323	**71**
19	0·326	0·328	0·331	0·334	0·337	0·339	**70**
20	0·342	0·345	0·347	0·350	0·353	0·356	**69**
21	0·358	0·361	0·364	0·367	0·369	0·372	**68**
22	0·375	0·377	0·380	0·383	0·385	0·388	**67**
23	0·391	0·393	0·396	0·399	0·401	0·404	**66**
24	0·407	0·409	0·412	0·415	0·417	0·420	**65**
25	0·423	0·425	0·428	0·431	0·433	0·436	**64**
26	0·438	0·441	0·444	0·446	0·449	0·451	**63**
27	0·454	0·457	0·459	0·462	0·464	0·467	**62**
28	0·469	0·472	0·475	0·477	0·480	0·482	**61**
29	0·485	0·487	0·490	0·492	0·495	0·497	**60**
30	0·500	0·503	0·505	0·508	0·510	0·513	**59**
31	0·515	0·518	0·520	0·522	0·525	0·527	**58**
32	0·530	0·532	0·535	0·537	0·540	0·542	**57**
33	0·545	0·547	0·550	0·552	0·554	0·557	**56**
34	0·559	0·562	0·564	0·566	0·569	0·571	**55**
35	0·574	0·576	0·578	0·581	0·583	0·585	**54**
36	0·588	0·590	0·592	0·595	0·597	0·599	**53**
37	0·602	0·604	0·606	0·609	0·611	0·613	**52**
38	0·616	0·618	0·620	0·623	0·625	0·627	**51**
39	0·629	0·632	0·634	0·636	0·638	0·641	**50**
40	0·643	0·645	0·647	0·649	0·652	0·654	**49**
41	0·656	0·658	0·660	0·663	0·665	0·667	**48**
42	0·669	0·671	0·673	0·676	0·678	0·680	**47**
43	0·682	0·684	0·686	0·688	0·690	0·693	**46**
44	0·695	0·697	0·699	0·701	0·703	0·705	**45**
45	0·707						**44**
	60′	50′	40′	30′	20′	10′	D
	Cosine						

TABLE OF NATURAL SINES AND TANGENTS (*continued*)

Deg.	Cosine						
	0′	10′	20′	30′	40′	50′	
0	1·000	1·000	1·000	1·000	1·000	1·000	**89**
1	1·000	1·000	1·000	1·000	1·000	0·999	**88**
2	0·999	0·999	0·999	0·999	0·999	0·999	**87**
3	0·999	0·998	0·998	0·998	0·998	0·998	**86**
4	0·998	0·997	0·997	0·997	0·997	0·996	**85**
5	0·996	0·996	0·996	0·995	0·995	0·995	**84**
6	0·995	0·994	0·994	0·994	0·993	0·993	**83**
7	0·993	0·992	0·992	0·991	0·991	0·991	**82**
8	0·990	0·990	0·989	0·989	0·989	0·988	**81**
9	0·988	0·987	0·987	0·986	0·986	0·985	**80**
10	0·985	0·984	0·984	0·983	0·983	0·982	**79**
11	0·982	0·981	0·981	0·980	0·979	0·979	**78**
12	0·978	0·978	0·977	0·976	0·976	0·975	**77**
13	0·974	0·974	0·973	0·972	0·972	0·971	**76**
14	0·970	0·970	0·969	0·968	0·967	0·967	**75**
15	0·966	0·965	0·964	0·964	0·963	0·962	**74**
16	0·961	0·960	0·960	0·959	0·958	0·957	**73**
17	0·956	0·955	0·955	0·954	0·953	0·952	**72**
18	0·951	0·950	0·949	0·948	0·947	0·946	**71**
19	0·946	0·945	0·944	0·943	0·942	0·941	**70**
20	0·940	0·939	0·938	0·937	0·936	0·935	**69**
21	0·934	0·933	0·931	0·930	0·929	0·928	**68**
22	0·927	0·926	0·925	0·924	0·923	0·922	**67**
23	0·921	0·919	0·918	0·917	0·916	0·915	**66**
24	0·914	0·912	0·911	0·910	0·909	0·908	**65**
25	0·906	0·905	0·904	0·903	0·901	0·900	**64**
26	0·899	0·898	0·896	0·895	0·894	0·892	**63**
27	0·891	0·890	0·888	0·887	0·886	0·884	**62**
28	0·883	0·882	0·880	0·879	0·877	0·876	**61**
29	0·875	0·873	0·872	0·870	0·869	0·867	**60**
30	0·866	0·865	0·863	0·862	0·860	0·859	**59**
31	0·857	0·856	0·854	0·853	0·851	0·850	**58**
32	0·848	0·847	0·845	0·843	0·842	0·840	**57**
33	0·839	0·837	0·835	0·834	0·832	0·831	**56**
34	0·829	0·827	0·826	0·824	0·822	0·821	**55**
35	0·819	0·817	0·816	0·814	0·812	0·811	**54**
36	0·809	0·807	0·806	0·804	0·802	0·800	**53**
37	0·799	0·797	0·795	0·793	0·792	0·790	**52**
38	0·788	0·786	0·784	0·783	0·781	0·779	**51**
39	0·777	0·775	0·773	0·772	0·770	0·768	**50**
40	0·766	0·764	0·762	0·760	0·759	0·757	**49**
41	0·755	0·753	0·751	0·749	0·747	0·745	**48**
42	0·743	0·741	0·739	0·737	0·735	0·733	**47**
43	0·731	0·729	0·727	0·725	0·723	0·721	**46**
44	0·719	0·717	0·715	0·713	0·711	0·709	**45**
45	0·707						**44**
	60′	50′	40′	30′	20′	10′	D
	Sine						

TABLE OF NATURAL SINES AND TANGENTS (*continued*)

Deg.	Tan						
	0′	10′	20′	30′	40′	50′	
0	0·000	0·003	0·006	0·009	0·012	0·015	**89**
1	0·017	0·020	0·023	0·026	0·029	0·032	**88**
2	0·035	0·038	0·041	0·044	0·047	0·049	**87**
3	0·052	0·055	0·058	0·061	0·064	0·067	**86**
4	0·070	0·073	0·076	0·079	0·082	0·085	**85**
5	0·087	0·090	0·093	0·096	0·099	0·102	**84**
6	0·105	0·108	0·111	0·114	0·117	0·120	**83**
7	0·123	0·126	0·129	0·132	0·135	0·138	**82**
8	0·141	0·144	0·146	0·149	0·152	0·155	**81**
9	0·158	0·161	0·164	0·167	0·170	0·173	**80**
10	0·176	0·179	0·182	0·185	0·188	0·191	**79**
11	0·194	0·197	0·200	0·203	0·206	0·210	**78**
12	0·213	0·216	0·219	0·222	0·225	0·228	**77**
13	0·231	0·234	0·237	0·240	0·243	0·246	**76**
14	0·249	0·252	0·256	0·259	0·262	0·265	**75**
15	0·268	0·271	0·274	0·277	0·280	0·284	**74**
16	0·287	0·290	0·293	0·296	0·299	0·303	**73**
17	0·306	0·309	0·312	0·315	0·318	0·322	**72**
18	0·325	0·328	0·331	0·335	0·338	0·341	**71**
19	0·344	0·348	0·351	0·354	0·357	0·361	**70**
20	0·364	0·367	0·371	0·374	0·377	0·381	**69**
21	0·384	0·387	0·391	0·394	0·397	0·401	**68**
22	0·404	0·407	0·411	0·414	0·418	0·421	**67**
23	0·424	0·428	0·431	0·435	0·438	0·442	**66**
24	0·445	0·449	0·452	0·456	0·459	0·463	**65**
25	0·466	0·470	0·473	0·477	0·481	0·484	**64**
26	0·488	0·491	0·495	0·499	0·502	0·506	**63**
27	0·510	0·513	0·517	0·521	0·524	0·528	**62**
28	0·532	0·535	0·539	0·543	0·547	0·551	**61**
29	0·554	0·558	0·562	0·566	0·570	0·573	**60**
30	0·577	0·581	0·585	0·589	0·593	0·597	**59**
31	0·601	0·605	0·609	0·613	0·617	0·621	**58**
32	0·625	0·629	0·633	0·637	0·641	0·645	**57**
33	0·649	0·654	0·658	0·662	0·666	0·670	**56**
34	0·675	0·679	0·683	0·687	0·692	0·696	**55**
35	0·700	0·705	0·709	0·713	0·718	0·722	**54**
36	0·727	0·731	0·735	0·740	0·744	0·749	**53**
37	0·754	0·758	0·763	0·767	0·772	0·777	**52**
38	0·781	0·786	0·791	0·795	0·800	0·805	**51**
39	0·810	0·815	0·819	0·824	0·829	0·834	**50**
40	0·839	0·844	0·849	0·854	0·859	0·864	**49**
41	0·869	0·874	0·880	0·885	0·890	0·895	**48**
42	0·900	0·906	0·911	0·916	0·922	0·927	**47**
43	0·933	0·938	0·943	0·949	0·955	0·960	**46**
44	0·966	0·971	0·977	0·983	0·988	0·994	**45**
45	1·000						**44**
	60′	50′	40′	30′	20′	10′	D
	cot						

TABLE OF NATURAL SINES AND TANGENTS (*continued*)

Deg.	Cot						
	0°	10′	20′	30′	40′	50′	
0	∞	343·8	171·9	114·6	85·94	68·75	**89**
1	57·29	49·10	42·96	38·19	34·37	31·24	**88**
2	28·64	26·43	24·54	22·90	21·47	20·21	**87**
3	19·08	18·07	17·17	16·35	15·60	14·92	**86**
4	14·30	13·73	13·20	12·71	12·25	11·83	**85**
5	11·43	11·06	10·71	10·39	10·08	9·788	**84**
6	9·514	9·255	9·010	8·777	8·556	8·345	**83**
7	8·144	7·953	7·770	7·596	7·429	7·269	**82**
8	7·115	6·968	6·827	6·691	6·561	6·435	**81**
9	6·314	6·197	6·084	5·976	5·871	5·769	**80**
10	5·671	5·576	5·485	5·396	5·309	5·226	**79**
11	5·145	5·066	4·989	4·915	4·843	4·773	**78**
12	4·705	4·638	4·574	4·511	4·449	4·390	**77**
13	4·331	4·275	4·219	4·165	4·113	4·061	**76**
14	4·011	3·962	3·914	3·867	3·821	3·776	**75**
15	3·732	3·689	3·647	3·606	3·566	3·526	**74**
16	3·487	3·450	3·412	3·376	3·340	3·305	**73**
17	3·271	3·237	3·204	3·172	3·140	3·108	**72**
18	3·078	3·047	3·018	2·989	2·960	2·932	**71**
19	2·904	2·877	2·850	2·824	2·798	2·773	**70**
20	2·747	2·723	2·699	2·675	2·651	2·628	**69**
21	2·605	2·583	2·560	2·539	2·517	2·496	**68**
22	2·475	2·455	2·434	2·414	2·394	2·375	**67**
23	2·356	2·337	2·318	2·300	2·282	2·264	**66**
24	2·246	2·229	2·211	2·194	2·177	2·161	**65**
25	2·145	2·128	2·112	2·097	2·081	2·066	**64**
26	2·050	2·035	2·020	2·006	1·991	1·977	**63**
27	1·963	1·949	1·935	1·921	1·907	1·894	**62**
28	1·881	1·868	1·855	1·842	1·829	1·816	**61**
29	1·804	1·792	1·780	1·767	1·756	1·744	**60**
30	1·732	1·720	1·709	1·698	1·686	1·675	**59**
31	1·664	1·653	1·643	1·632	1·621	1·611	**58**
32	1·600	1·590	1·580	1·570	1·560	1·550	**57**
33	1·540	1·530	1·520	1·511	1·501	1·492	**56**
34	1·483	1·473	1·464	1·455	1·446	1·437	**55**
35	1·428	1·419	1·411	1·402	1·393	1·385	**54**
36	1·376	1·368	1·360	1·351	1·343	1·335	**53**
37	1·327	1·319	1·311	1·303	1·295	1·288	**52**
38	1·280	1·272	1·265	1·257	1·250	1·242	**51**
39	1·235	1·228	1·220	1·213	1·206	1·199	**50**
40	1·192	1·185	1·178	1·171	1·164	1·157	**49**
41	1·150	1·144	1·137	1·130	1·124	1·117	**48**
42	1·111	1·104	1·098	1·091	1·085	1·079	**47**
43	1·072	1·066	1·060	1·054	1·048	1·042	**46**
44	1·036	1·030	1·024	1·018	1·012	1·006	**45**
45	1·000						**44**
	60′	50′	40′	30′	20′	10′	D
	Tan						

THE SLIDE-RULE

Structure, Direction for Use

There are different kinds of slide-rules, from the ordinary Faber 10 in. to the spiral, such as Fuller's, which is equivalent to a straight rule of several feet in length. There are also slide-rules made for special calculations.

The main body of the 10 in. rule is known as the stock, and this carries a slide, the upper surface of which is level with that of the stock. On the face of the rule are four scales; that on the upper side of the stock is called *A*, that on the top of the slide *B*, that on the lower side of the slide *C*, and that on the lower side of the stock *D*. The divisions on *A* and *B* correspond; those on *C* and *D* also correspond, but are twice as long as the scales on *A* and *B*. A small metal frame enclosing a glass in the middle of which a vertical line has been drawn, is known as a runner, covering all four scales and movable freely backward and forward, which enables one to bring into line the divisions on the slide with those of the stock.

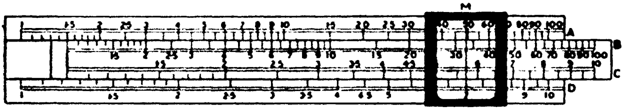


FIG. 3. SLIDE-RULE

The primary divisions of the first half of *A* and *B* from left to right are numbered 1 to 10, while those on the second or right-hand half are numbered 10 to 100. The primary divisions are not equally spaced as on a foot rule, but gradually diminish toward the right, because the divisions represent the logarithm of the number by which it is distinguished. The subdivisions of *D* and *C* are divided on the same principle. In the diagram the smaller divisions seen on the actual scale are omitted for the sake of clearness. The value assigned to the primary divisions in any calculation corresponds to the intermediate divisions also. If the left index figure is taken as 10 instead of 1, all the other divisions are increased ten times and will read 20, 30, 40, etc., instead of 2, 3, 4, etc. Or, if the left index figure is taken as 0·1, the other primary figures will read 0·2, 0·3, 0·4, etc.

Rule for multiplication. "Set the index of *B* scale to one of the factors on *A*, and over the other factor on *B* read the product on *A*."

Division. This is the reverse of multiplication. "Set the divisor on *B* under the dividend on *A*, and read the quotient on *A* over the index of *B*."

To convert a fraction to a decimal. "Place the numerator on *C* to the denominator on *D*, and read the answer over 1 on *D*."

To convert a decimal to a vulgar fraction. "The decimal part on *C* is set to the index of *D*, then opposite the numerator on *C* is the corresponding denominator of the fraction on *D*."

Proportion. "Set the first term of the proportion on the *C* scale to the second term on *D*, and opposite the third term on *C* read the fourth term on *D* scale."

Squares. "Set the runner to the given number on *D*, and read off the required square on *A* under the runner."

Square Root. This is the inverse of obtaining the square. "Set the runner to the given number on *A* and read off the required square root on *D* under the runner."

Cube. "Bring the right or left-hand index of *C* to the given number on *D*, and over the same number on the left-hand *B* scale read the required cube on *A*."

Cube Root. This is the inverse of the former. "Move the slide, either from right to left, or from left to right, until under the given number on *A* is found a number on the left-hand of *B* scale identical with the number which is simultaneously found on *D*, under the right- or left-hand index of *C*. This number is the required cube root."

On the reverse side of the slide are three scales marked *S*, *L*, and *T*. *S* is the scale for sines, *T* the scale for tangents, and *L* is the scale for logarithms.

BRITISH WEIGHTS AND MEASURES

AVOIRDUPOIS WEIGHT

Systematic Name	Dram (dr)	Ounce (oz)	Pound (lb)	Stone (st)	Quarter (qr)	Cental, or New Hundred-weight	Hundred-weight (cwt)	Ton	Metric equivalent (grammes)
Dram (dr)	1	0·0625	0·0039	0·000279	0·000139	0·000039	0·000035	0·00000174	1·771846
Ounce (oz)	16	1	0·0625	0·00446	0·00223	0·000625	0·000558	0·000028	28·34954
Pound (lb)	256	16	1	0·0714	0·0357	0·01	0·00893	0·000447	453·5926
Stone (st)	3584	224	14	1	0·5	0·14	0·125	0·00625	6350·302
Quarter (qr)	7168	448	28	2	1	0·28	0·25	0·0125	12,700·60
Cental or New Hundredweight	25,600	1600	100	7·1428	3·5714	1	0·893	0·0447	45,359·25
Hundredweight (cwt)	28,672	1792	112	8	4	1·12	1	0·05	50,802·42
Ton	573,440	35,840	2240	160	80	22·40	20	1	1,016,048

1 dram = 27·346 troy grains. 1 pound = 7000·576 troy grains.
1 sack of coal = 2 hundredweight, or 224 lb; 10 sacks or 2240 lb go to the ton.
A pack of wool weighs 240 lb.
A truss of straw weighs 36 lb; a truss of old hay 56 lb, and of new, 60 lb.
A load consists of 36 trusses.
A short ton is 2000 lb.

Used in trade and commerce for larger and coarser sorts of goods.

TROY WEIGHT

Systematic Name	Grain (gr)	Penny-weight (dwt)	Ounce (oz)	Pound (lb)	Metric Equivalent (grammes)
Grain (gr)	1	0·04167	0·002083	0·0001736	0·0648
Pennyweight (dwt)	24	1	0·05	0·004167	1·555
Ounce (oz)	480	20	1	0·0833	31·1035
Pound (lb)	5760	240	12	1	373·242

1 gr troy, avoirdupois and apothecaries are equal.

Troy weight is used for gold and silver. Perfectly pure gold is worth £4 4s. 11·45d., or $20·67183 per troy oz. Fine gold is said to be 24-carat. Standard gold is 22-carat alloyed with two parts of some other metal. Standard silver is 11 oz 2 dwt fine to 18 dwt alloy.

The jeweller's carat is 3·17 gr in England, 3·18 gr in France, 3·0 gr in Holland, and 3·2 gr in the United States; the carat is divided into 4 jeweller's grains, and the jewellery ounce into 151½ carats. The international carat weighs 3·168 gr or 205 mg.

APOTHECARIES' WEIGHT

Systematic Name	Grain (gr)	Scruple (℈)	Dram (℥)	Ounce (ʒ)	Pound (lb)
Grain (gr)	1	0·05	0·016	0·002083	0·00017361
Scruple (℈)	20	1	0·3	0·0416	0·003472
Dram (ʒ)	60	3	1	0·125	0·010416
Ounce (℥)	480	24	8	1	0·083
Pound (lb)	5760	288	96	12	1

In troy and apothecaries' weights the grain, ounce, and pound are the same. Used for medical prescriptions. Drugs are bought and sold in quantities by avoirdupois.

BRITISH IMPERIAL MEASURE OF CAPACITY

Systematic Name	Gill	Pint	Quart	Pottle	Gallon	Peck	Bushel	Coomb	Quarter	Cu ft	Litres
Gill . .	1	0·25	0·125	0·0625	0·03125	0·015625	0·003906	0·0009765	0·0004882	0·005	0·1419
Pint . .	4	1	0·5	0·25	0·125	0·0625	0·01562	0·003905	0·00195	0·02	0·5676
Quart .	8	2	1	0·5	0·25	0·125	0·03125	0·0078125	0·003906	0·04	1·1352
Pottle .	16	4	2	1	0·5	0·25	0·0625	0·015625	0·007812	0·08	2·2704
Gallon .	32	8	4	2	1	0·5	0·125	0·03125	0·0156	0·1604	4·541
Peck .	64	16	8	4	2	1	0·25	0·0625	0·03125	0·3208	9·082
Bushel .	256	64	32	16	8	4	1	0·25	0·125	1·283	36·32816
Coomb .	1024	256	128	64	32	16	4	1	0·5	5·132	145·31264
Quarter .	2048	512	256	128	64	32	8	2	1	10·264	290·625

	Gal	Qt	Pt
Firkin or quarter barrel . . .	9	36	72
Kilderkin or half barrel . . .	18	72	144
Barrel	36	144	288
Hogshead = 1½ barrels . . .	54	216	432
Puncheon	72	288	576
Butt = 3 barrels	108	432	864

10 lb. distilled water = 277¼ cu in. = 0·16 cu ft = 1 gal. 1 ton of water = 25·9 cu ft.
1 cu ft water = 62·335 lb. 36 cu ft = 1 ton.

Apothecaries' Fluid Measure

60 minims (♏) = 1 fluid drachm (fʒ)
8 drachms = 1 ounce (f℥)
20 ounces = 1 pint (O)
8 pints = 1 gallon (gal)
1 drop = 1 grain
60 drops = 1 drachm
4 drachms = 1 tablespoonful
2 ounces = 1 wineglassful
3 ounces = 1 teacupful
70,000 grains = 1 gallon

Liquid Measure, U.S. only

4 gills = 1 pint = 28·875 cu in.
2 pints = 1 quart = 57·750 ,,
4 quarts = 1 gallon = 231·000 ,,
63 gallons = 1 hogshead
2 hogsheads = 1 pipe or butt
2 pipes = 1 tun

Dry Measure, U.S. only

2 pints = 1 quart = 67·2006 cu in.
4 quarts = 1 gallon = 268·8025 ,,
2 gallons = 1 peck = 537·6050 ,,
4 pecks = 1 struck bushel = 2150·4200 ,,

LONG MEASURE

Systematic Name	Inch (in.)	Foot (ft)	Yard (yd)	Rod, Pole, or Perch (p)	Chain (ch)	Furlong (fur)	Mile (m)	Metric Equivalent (metres)
Inch (in.)	1	0·083	0·02778	0·005	0·00126	0·000126	0·0000158	0·0254
Foot (ft)	12	1	0·333	0·0606	0·01515	0·00151	0·0001894	0·3048
Yard (yd)	36	3	1	0·182	0·045	0·00454	0·000568	0·9144
Rod, pole or perch (p)	198	16½	5½	1	0·25	0·0091	0·001136	1·8287
Chain (ch)	792	66	22	4	1	0·025	0·003125	5·0291
Furlong (fur)	7920	660	220	40	10	1	0·125	201·16
Mile (m)	63,360	5280	1,760	320	80	8	1	1609·315

1 point = $\frac{1}{72}$ of an inch; 1 line = $\frac{1}{12}$ of an inch; 1 link = 7·92 inches; 100 links = 1 Gunter's chain; 1 fathom = 6 feet; 1 league = 3 miles; 1 degree = 69⅛ miles = 60 nautical knots or geographical miles; 1 geographical mile = 6082·66 feet; 80 Gunter's chains = 1 mile.

Square of Surface Measure, for Land, Boards, Painting, Paving, Plastering, etc.

	Sq in.	Sq ft	Sq yd	Sq p	Sq ch	Ro	Ac	Value in sq metres
1 Square foot .	**144**	1	0·111	0·00367309	0·000229568	0·000091827	0·0000229568	0·092894
1 Square yard .	1296	**9**	1	0·0330578	0·00206611	0·00082644	0·000206611	0·836046
1 Square pole or rod	39,204	272¼	**30¼**	1	0·0625	0 025	0·00625	25·302915
1 Square chain	627,264	4356	484	16	1	0·4	0·1	404·8466
1 Rood . .	1,568,160	10,890	1210	**40**	2½	1	0·25	1012·1166
1 Acre . .	6,272,640	43,560	4840	160	**10**	**4**	1	4048·4664
1 Square mile .	4,014,489,600	27,878,400	3,097,600	102,400	6400	2560	**640**	2,591,018·49

One square mile = 1 section = 640 acres; 36 square miles (a square with a six-mile side) = 1 township; 10 square chains or 100,000 square links = 1 acre; a "hide" of land = 100 acres. In measuring flooring roofing, plastering, etc., a "square" = 100 square feet.

CUBIC OR SOLID MEASURE, FOR TIMBER, STONE, BOXES, PACKAGES, ROOMS, ETC.

	Cu in.	Cu ft	Cu yd	Value in cu metre
1 Cubic foot .	**1728**	1	0·037	0·02831
1 Cubic yard .	46,656	**27**	1	0·76437

1 ton or load = 40 cubic feet of rough or 50 cubic feet of hewn timber.

1 shipping ton = 42 cubic feet of timber; 1 ton shipping = 40 cubic feet.

1 stack of wood = 108 cubic feet; 1 cord of wood = 128 cubic feet = a pile 4 ft high, 4 ft wide, and 8 ft long; 1 cord foot is a foot in length of such a pile.

1 register ton = 100 cubic feet; 1 cubic yard of ordinary earth is called a load; 1 board foot is 1 ft long, 1 ft wide, and 1 in. thick; 12 board feet = 1 cubic foot; a barrel's bulk = 50 cubic feet; 1 cubic foot of water = about 6¼ gallons = 62·321 lb; 1 gallon = 277¼ cu in. = 0·16 cu ft.

1 ton weight of shingle = about 23 cubic feet
,, ,, ,, coarse gravel = ,, 19 ,, ,,
,, ,, pit sand = ,, 22 ,, ,,
,, ,, clay or marl = ,, 18 ,, ,,

Calculate 100 tons of coal per inch per acre which allows about 25 per cent for loss of every kind.

CARPENTERS', BRICKLAYERS', AND BUILDERS' MEASUREMENTS

Stock bricks . .	8¾ inches	× 4¼	× 2¾
Welsh fire-bricks. .	9 ,,	× 4½	× 2¾
Dutch clinkers . .	9¼ ,,	× 3	× 1½

500 bricks = a load. 5 courses of bricks in the height of a wall are called a foot. 1 rod of brickwork = 306 cubic feet = 11⅓ cubic yards, and contains about 4500 bricks with about 75 cubic feet of mortar.

Measure of Timber

Boards of 7 in. wide = battens
,, 9 ,, = deals
,, 12 ,, = planks

100 superficial feet of planking = 1 square
120 deals = 1 hundred
50 cu ft squared timber = 1 load
40 cu ft unhewn timber = 1 load
600 superficial feet of inch planking = 1 load

Props are bought and sold per 72 running feet, the price depending on the diameter.

To calculate the cubic contents of large round timber, multiply the length by the square of the quarter girth.

Sizes of Slates

Princesses	24 × 14 in.	Ladies	16 × 8 in.
Duchesses	24 × 12 ,,	Ditto	15 × 8 ,,
Marchionesses	22 × 12 ,,	Ditto	14 × 12 ,,
Countesses	20 × 10 ,,	Ditto	14 × 8 ,,
Viscountesses	18 × 10 ,,	Plantations	13 × 11 ,,
Ditto	18 × 9 ,,	Doubles	13 × 18 ,,
Ladies	18 × 10 ,,	Ditto	12 × 20 ,,

Angular Measure

60 seconds = 1 minute
60 minutes = 1 degree
30 degrees = 1 sign
90 degrees = 1 quadrant
4 quadrants or 360° = 1 circumference or great circle

PAPER

1 ream = 20 quires = 480 sheets
1 quire = 24 sheets

Drawing Paper

Cap	13 × 16 inches
Demy	20 × 15 ,,
Medium	22 × 17 ,,
Royal	24 × 19 ,,
Super Royal	27 × 19 ,,
Imperial	30 × 21 ,,
Elephant	28 × 22 ,,
Columbier	34 × 23 ,,
Atlas	33 × 26 ,,
Theorem	34 × 28 ,,
Double Elephant	40 × 26 ,,
Antiquarian	52 × 31 ,,
Emperor	40 × 60 ,,
Uncle Sam	48 × 120 ,,

Metric Weights and Measures

The **Metrical system** is based upon the length of the fourth part of a terrestrial meridian. The ten-millionth part of this arc was chosen as the unit of measures of length, and called a *Mètre*. The cube of a tenth part of the metre was adopted as the unit of capacity, and denominated a *Litre*. The weight of a litre of distilled water at its greatest density was called a *Kilogramme*, of which the thousandth part, or *Gramme*, was adopted as the unit of weight. The multiples of these, proceeding in decimal progression, are distinguished by the employment of the prefixes *deca*, *hecto*, *kilo*, and *myria*, from the Greek, and the subdivisions by *deci*, *centi*, and *milli*, from the Latin.

MEASURES OF LENGTH (UNIT METRE)

EQUAL TO	Inches	Feet	Yards	Fathoms	Miles
Millimetre	0·03937	0·003281	0·0010936	0·0005468	0·0000006
Centimetre	0·39371	0·032809	0·0109363	0·0054682	0·0000062
Decimetre	3·93708	0·328090	0·1093633	0·0546816	0·0000621
METRE	39·37079	3·280899	1·0936331	0·5468165	0·0006214
Decametre	393·70790	32·808992	10·9363306	5·4681653	0·0062138
Hectometre	3,937·07900	328·089917	109·3633056	54·6816528	0·0621382
Kilometre	39,370·79000	3,280·899167	1,093·6330556	546·8165278	0·6213824
Myriametre	393,707·90000	32,808·991667	10,936·3305556	5,468·1652778	6·2138242

CUBIC, OR MEASURES OF CAPACITY (UNIT LITRE)

EQUAL TO	Cubic inches	Cubic feet	Pints	Gallons	Bushels
Millilitre, or cubic centimetre	0·06103	0·000035	0·00176	0·0002201	0·0000275
Centilitre, 10 cubic centimetres	0·61027	0·000353	0·01761	0·0022010	0·0002751
Decilitre, 100 cubic centimetres	6·10271	0·003532	0·17608	0·0220097	0·0027512
LITRE, or cubic Decimetre	61·02705	0·035317	1·76077	0·2200967	0·0275121
Decalitre or Centistere	610·27052	0·353166	17·60773	2·2009668	0·2751208
Hectolitre or Decistere	6,102·70515	3·531658	176·07734	22·0096677	2·7512085
Kilolitre, or Stere, or cubic metre	61,027·05152	35·316581	1,760·77341	220·0966767	27·5120846
Myrialitre, or Decastere	610,270·51519	353·165807	17,607·73414	2,200·9667675	275·1208459

MEASURES OF WEIGHT (UNIT GRAMME)

EQUAL TO	Grains	Troy Oz	Avoirdupois Lb	Cwt (112 lb)	Tons (20 cwt)
Milligramme	0·01543	0·000032	0·0000022	0·0000000	0·0000000
Centigramme	0·15432	0·000322	0·0000220	0·0000002	0·0000000
Decigramme	1·54323	0·003215	0·0002205	0·0000020	0·0000001
GRAMME	15·43235	0·032151	0·0022046	0·0000197	0·0000010
Decagramme	154·32349	0·321507	0·0220462	0·0001968	0·0000098

MEASURES OF WEIGHT (UNIT GRAMME) (*continued*)

EQUAL TO	Grains	Troy Oz	Avoirdupois Lb	Cwt (112 lb)	Tons (20 cwt)
Hectogramme	1,543·23488	3·215073	0·2204621	0·0019684	0·0000984
Kilogramme	15,432·34880	32·150727	2·2046213	0·0196841	0·0009842
Myriagramme	154,323·48800	321·507267	22·0462126	0·1968412	0·0098421

SQUARE, OR MEASURE OF SURFACE (UNIT ARE)

EQUAL TO	Sq Feet	Sq Yards	Sq Perches	Sq Roods	Acres
Centiare, or square metre . .	10·764299	1·196033	0·0395383	0·0009885	0·0002471
ARE, or 100 square metres . .	1,076·429934	119·603326	3·9538290	0·0988457	0·0247114
Hectare, or 10,000 square metres .	107,642·993419	11,960·332602	395·3828059	9·8845724	2·4711431

Millimetres × 0·03937 = inches.
Millimetres ÷ 25·4 = inches.
Centimetres × 0·3937 = inches.
Centimetres ÷ 2·54 = inches.
Metres × 39·37 = inches. (Act Congress.)
Metres × 3·281 = feet.
Metres × 1·094 = yards.
Kilometres × 0·621 = miles.
Kilometres ÷ 1·6093 = miles.
Kilometres × 3280·8693 = feet.
Square Millimetres × 0·00155 = sq inches.
Square Millimetres ÷ 645·1 = sq inches.
Square Centimetres × 0·155 = sq inches.
Square Centimetres ÷ 6·451 = sq inches.
Square Metres × 10·764 = sq feet.
Square Kilometres × 247·1 = acres.
Hectare × 2·471 = acres.
Cubic Centimetres ÷ 16·383 = cubic inches.
Cubic Centimetres ÷ 3·69 = fl drams (U.S.P.).
Cubic Centimetre ÷ 29·57 = fluid oz (U.S.P.).
Cubic Metres × 35·315 = cubic feet.
Cubic Metres × 1·308 = cubic yards.
Cubic Metres × 264·2 = gallons (231 cu in.).
Litres × 61·022 = cubic inches. (Act Congress.)
Litres × 33·84 = fluid ounces (U.S. PHAR.).
Litres × 0·2642 = gallons (231 cu in.).
Litres ÷ 3·78 = gallons (231 cu in.).

Litres ÷ 28,316 = cubic feet.
Hectolitres × 3·531 = cubic feet.
Hectolitres × 2·84 = bushels (2150·42 cu in.)
Hectolitres × 0·131 = cubic yards.
Hectolitres ÷ 26·42 = gallons (231 cu in.).
Grammes × 15·432 = grains. (Act Congress.)
Grammes ÷ 981 = dynes.
Grammes (water) ÷ 29·57 = fluid ounces.
Grammes ÷ 28·35 = ounces avoirdupois.
Grammes per cu. cent. ÷ 27·7 = lb per cu in.
Joule × 0·7373 = foot pounds.
Kilogrammes × 2·2046 = pounds.
Kilogrammes × 35·3 = ounces avoirdupois.
Kilogrammes ÷ 907·2 = tons (2000 lb).
Kilogrammes per sq. cent × 14·223 = lb per sq in.
Kilogram-metres × 7·233 = foot lb.
Kilogramme per Metre × 0·672 = lb per foot.
Kilogramme per Cu Metre × 0·062 = lb per cu ft.
Kilogrammes per Cheval × 2·235 = lb per h.p.
Kilowatts × 1·34 = Horse-power.
Watts ÷ 746 = Horse-power.
Watts × 0·7373 = foot-pounds per second.
Calorie × 3·968 = B.Th.U.
Cheval vapeur × 0·9863 = Horse-power.
(Centigrade × 1·8) + 32 = degrees Fahrenheit.
Gravity Paris = 980·94 centimetres per sec.

WEIGHTS AND MEASURES OF FOREIGN COUNTRIES

Many countries, e.g. France and Germany, have adopted the metric system, while others retain their special weights and measures.

Russian

A funt is equivalent to 0·4095124 kg or 0·9028 lb avdp.

1 funt = 96 zolotniks or 32 loths.

1 zolotnik = 96 doli.

40 funts = 1 pood.

Arshin = 0·71120 m or 28 in. is the basis of length.

1 arshin contains 16 verchoks.

3 arshins = 1 sagene.

500 sagenes = 1 verst.

1 dessiatine is the unit for land measurement. It is equal to 2·400 sq sagenes = 2·6997 acres.

1 tchetvert is the unit for dry measure = 2·09908 hectolitres.

The tchetvert is divided into 8 tchetveriks.

1 tchetverik = 8 garnetz.

1 last = 12 tchetverts.

Spanish-American

Arroba = 25·4 lb avdp. = 11·42 kg = ¼ quintal. In Spain an arroba = 1 cántara.

Tonelada = 20 quintals = 921·8 kg or 2032·2 lb avdp.

A vara = 2·75 ft (Peru), 2·841 ft (Argentina), 2·782 ft (Spain).

Estaca (Peru) = 167 metres square, or 27,889 sq m.

Fanega (Peru) = 1·65 acre (varying locally); also 141·43 lb avdp. = 64·14 kg.

Pertenencia, or mining claim (Mexico) = 167·6 m horizontal on strike of vein multiplied by the width of 94·28 to 167·6 m according to the dip.

Weights used for silver and gold in Peru: 3 granos = 1 tomin. 12 tomines = 1 adarme; 16 adarmes = 1 onza; 8 onzas = 1 marco; 2 marcos = 1 libra (equal to 1·014 lb in Spain and Mexico); 1 marco = 50 castellanos = 230·07 g = 3550·54 gr = 0·507 lb avdp.; 1 onza = 1·02 oz troy; 1 oz troy = 6·755 castellanos. In Mexico the silver content of an ore is expressed in marcos per carga. 1 marco per carga = 6·571 oz troy per ton.

Chinese

Length

Chinese		Metric
1 Hao	=	0·0032 cm
1 Li	=	0·032
1 Fen	=	0·32
1 Ts'un	=	3·2
1 Ch'ih	=	0·32 m
1 Pu	=	1·6
1 Chang	=	3·2
1 Li	=	576·0

Surface

Chinese		Metric
1 Hao	=	0·006144 Are
1 Li	=	0·06144
1 Fen	=	0·6144
1 Fang pu	=	0·0256
1 Mu	=	6·144 Ares
1 Ch'ing	=	614·4

Capacity

1 Shao	=	0·0104 l
1 Ko	=	0·1035
1 Sheng	=	1·0335
1 Tou	=	10·355
1 Hu	=	51·7734
1 Tan or Picul	=	103·5469

Weight

1 Hao	=	0·0037301 g
1 Li	=	0·037301
1 Fen	=	0·37301
1 Ch'ien	=	3·7301
1 Liang	=	37·301
1 Chin or Catty	=	596·816

Japanese

Weight. A Kwan is the unit of mass and is equal to 3·75 kg or 8·267 lb avdp. It is divided into 1000 mommes.

1 Momme = 10 Funs
1 Fun = 10 Rins
1 Rin = 10 Mos
1 Mo = 10 Shis

Length. A Shaku is the unit of length = 0·30 metres. It is decimally divided into sun, bu, rin, mo, and shi. The multiples of the shaku are: 6 Shaku = 1 Ken; 60 Ken = 1 Cho; 36 Cho = 1 Ri.

Square or Land Measure. The Shaku = 0·00033 are (metric) or 0·988 sq ft.

10 Shaku = 1 Go
10 Gos = 1 Bu
30 Bu = 1 Tan Se
10 Se = 1 Tan
10 Tan = 1 Cho

Capacity. The shaku is equal to 0·01804 litres.

10 Shakus = 1 Go
10 Gos = 1 Sho
10 Shos = 1 To
10 Tos = 1 Koku

MATHEMATICAL TABLES

TABLE OF FRACTIONS REDUCED TO DECIMALS

$\frac{1}{64}$	0·015625	$\frac{17}{64}$	0·265625	$\frac{33}{64}$	0·515625	$\frac{49}{64}$	0·765625
$\frac{1}{32}$	0·03125	$\frac{9}{32}$	0·28125	$\frac{17}{32}$	0·53125	$\frac{25}{32}$	0·78125
$\frac{3}{64}$	0·046875	$\frac{19}{64}$	0·296875	$\frac{35}{64}$	0·546875	$\frac{51}{64}$	0·796875
$\frac{1}{16}$	0·0625	$\frac{5}{16}$	0·3125	$\frac{9}{16}$	0·5625	$\frac{13}{16}$	0·8125
$\frac{5}{64}$	0·078125	$\frac{21}{64}$	0·328125	$\frac{37}{64}$	0·578125	$\frac{53}{64}$	0·828125
$\frac{3}{32}$	0·09375	$\frac{11}{32}$	0·34375	$\frac{19}{32}$	0·59375	$\frac{27}{32}$	0·84375
$\frac{7}{64}$	0·109375	$\frac{23}{64}$	0·359375	$\frac{39}{64}$	0·609375	$\frac{55}{64}$	0·859375
$\frac{1}{8}$	0·125	$\frac{3}{8}$	0·375	$\frac{5}{8}$	0·625	$\frac{7}{8}$	0·875
$\frac{9}{64}$	0·140625	$\frac{25}{64}$	0·390625	$\frac{41}{64}$	0·640625	$\frac{57}{64}$	0·890625
$\frac{5}{32}$	0·15625	$\frac{13}{32}$	0·40625	$\frac{21}{32}$	0·65625	$\frac{29}{32}$	0·90625
$\frac{11}{64}$	0·171875	$\frac{27}{64}$	0·421875	$\frac{43}{64}$	0·671875	$\frac{59}{64}$	0·921875
$\frac{3}{16}$	0·1875	$\frac{7}{16}$	0·4375	$\frac{11}{16}$	0·6875	$\frac{15}{16}$	0·9375
$\frac{13}{64}$	0·203125	$\frac{29}{64}$	0·453125	$\frac{45}{64}$	0·703125	$\frac{61}{64}$	0·953125
$\frac{7}{32}$	0·21875	$\frac{15}{32}$	0·46875	$\frac{23}{32}$	0·71875	$\frac{31}{32}$	0·96875
$\frac{15}{64}$	0·234375	$\frac{31}{64}$	0·484375	$\frac{47}{64}$	0·734375	$\frac{63}{64}$	0·984375
$\frac{1}{4}$	0·25	$\frac{1}{2}$	0·5	$\frac{3}{4}$	0·75	1	1·0000

To find the area of an irregular polygon. Divide it into triangles, find the area of each triangle, and add them together. The sum will be the area of the polygon.

TABLE OF REGULAR POLYGONS WHOSE SIDES ARE UNITY

No. of Sides	Name of Polygon	Areas	Outer Radii	Angles Contained Between Two Sides	Angle at Centre of Circle
3	**Equilateral triangle**	0·4330	0·5774	60°	120°
4	**Square** . . .	1·0000	0·7071	90°	90°
5	**Pentagon** . .	1·7205	0·8507	108°	72°
6	**Hexagon** . .	2·5981	1·0000	120°	60°
7	**Heptagon** . .	3·6339	1·1524	128° 34′ 17″ +	51° 25′ 43″ −
8	**Octagon** . .	4·8284	1·3066	135°	45°
9	**Nonagon** . .	6·1818	1·4619	140°	40°
10	**Decagon** . .	7·6942	1·6180	144°	36°
11	**Undecagon** . .	9·3656	1·7747	147° 16′ 22″ −	32° 43′ 38″ +
12	**Dodecagon** . .	11·1962	1·9319	150°	30°

TABLE OF POWERS (ROOT, CIRCUMFERENCE, ARC, SQUARE, CUBE, SQUARE ROOT, AND CUBE ROOT)

To find the $\sqrt{n}$ of a number which is wholly decimal. If the number is not separable into twos, add another cipher to make it so. Then begin at the first numerical figure, and including it assume the number to be a whole one. In the table find the number nearest the assumed one, take out its tabular square root, move the decimal point of this tabular root to the left half as many places as the finally modified decimal number has figures.

EXAMPLE. $\sqrt{0·002}$? (1) 0·002; (2) 0·00·20; (3) $\sqrt{20} = 4·47$; (4) move decimal point two places; 0·0447 *Ans.* This rule is good up to three numerals inclusive.

To find the $\sqrt[3]{n}$ of a number which is wholly decimal. Proceed as in the former case for $\sqrt{n}$, only divide the figures into threes and point off to the left $\frac{1}{3}$ as many places as the finally modified decimal number has figures.

EXAMPLE. $\sqrt[3]{0·002}$? (1) 0·002; (2) $\sqrt[3]{2} = 1·26$; (3) 0·126 *Ans.*

To find the square or cube of any whole number ending with ciphers, (1) omit all the final ciphers; (2) take from the table the square or cube (as the case may be) of the rest of the number; (3) to this square add twice as many ciphers as there were final ciphers in the original number. To the cube add three times as many as the original number.

EXAMPLE. $28·000^2$? (1) $28^2 = 784$; (2) add six ciphers: 784,000,000 *Ans.*

EXAMPLE. $28·000^3$? (1) $28^3 = 21,952$; (2) add nine ciphers: 21,952,000,000,000 *Ans.*

n	$n\pi$	$n^2\frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
1·0	3·142	0·7854	1·000	1·000	1·0000	1·0000
1·1	3·456	0·9503	1·210	1·331	1·0488	1·0323
1·2	3·770	1·1310	1·440	1·728	1·0955	1·0627
1·3	4·084	1·3273	1·690	2·197	1·1402	1·0914
1·4	4·398	1·5394	1·960	2·744	1·1832	1·1187
1·5	4·712	1·7672	2·250	3·375	1·2247	1·1447
1·6	5·027	2·0106	2·560	4·096	1·2649	1·1696
1·7	5·341	2·2698	2·890	4·913	1·3038	1·1935
1·8	5·655	2·5447	3·240	5·832	1·3416	1·2164
1·9	5·969	2·8353	3·610	6·859	1·3784	1·2386
2·0	6·283	3·1416	4·000	8·000	1·4142	1·2599
2·1	6·597	3·4636	4·410	9·261	1·4491	1·2806

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
2·2	6·912	3·8013	4·840	10·648	1·4832	1·3006
2·3	7·226	4·1548	5·290	12·167	1·5166	1·3200
2·4	7·540	4·5239	5·760	13·824	1·5492	1·3389
2·5	7·854	4·9087	6·250	15·624	1·5811	1·3572
2·6	8·168	5·3093	5·760	17·576	1·6125	1·3751
2·7	8·482	5·7256	7·290	19·683	1·6432	1·3925
2·8	8·797	6·1575	7·840	21·952	1·6733	1·4095
2·9	9·111	6·6052	8·410	24·389	1·7029	1·4260
3·0	9·425	7·0686	9·00	27·000	1·7321	1·4422
3·1	9·739	7·5477	9·61	29·791	1·7607	1·4581
3·2	10·053	8·0425	10·24	32·768	1·7889	1·4736
3·3	10·367	8·5530	10·89	35·937	1·8166	1·4888
3·4	10·681	9·0792	11·56	39·304	1·8439	1·5037
3·5	10·996	9·6211	12·25	42·875	1·8708	1·5183
3·6	11·310	10·179	12·96	46·656	1·8974	1·5326
3·7	11·624	10·752	13·69	50·653	1·9235	1·5467
3·8	11·938	11·341	14·44	54·872	1·9494	1·5605
3·9	12·252	11·946	15·21	59·319	1·9748	1·5741
4·0	12·566	12·566	16·00	64·000	2·0000	1·5874
4·1	12·881	13·203	16·81	68·921	2·0249	1·6005
4·2	13·195	13·854	17·64	74·088	2·0494	1·6134
4·3	13·509	14·522	18·49	79·507	2·0736	1·6261
4·4	13·823	15·205	19·36	85·184	2·0976	1·6386
4·5	14·137	15·904	20·25	91·125	2·1213	1·6510
4·6	14·451	16·619	21·16	97·336	2·1448	1·6631
4·7	14·765	17·349	22·09	103·823	2·1680	1·6751
4·8	15·080	18·096	23·04	110·592	2·1909	1·6869
4·9	15·394	18·857	24·01	117·649	2·2136	1·6985
5·0	15·708	19·6350	25·00	125·000	2·2361	1·7100
5·1	16·022	20·4282	26·01	132·651	2·2583	1·7213
5·2	16·336	21·2372	27·04	140·608	2·2804	1·7325
5·3	16·650	22·0618	28·09	148·877	2·3022	1·7435
5·4	16·965	22·9022	29·16	157·464	2·3238	1·7544
5·5	17·279	23·7583	30·25	166·375	2·3452	1·7652
5·6	17·593	24·6301	31·36	175·616	2·3664	1·7758
5·7	17·907	25·5176	32·49	185·193	2·3875	1·7863
5·8	18·221	26·4208	33·64	195·112	2·4083	1·7967
5·9	18·535	27·3397	34·81	205·379	2·4290	1·8070
6·0	18·850	28·2743	36·00	216·000	2·4495	1·8171
6·1	19·174	29·2247	37·21	226·981	2·4698	1·8272
6·2	19·478	30·1907	38·44	238·328	2·4900	1·8371

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2\frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
6·3	19·792	31·1725	39·69	250·047	2·5100	1·8469
6·4	20·106	32·1699	40·96	262·144	2·5298	1·8566
6·5	20·420	33·1831	42·25	274·625	2·5495	1·8663
6·6	20·735	34·2119	43·56	287·496	2·5690	1·8758
6·7	21·049	35·2565	44·89	300·763	2·5884	1·8852
6·8	21·363	36·3168	46·24	314·432	2·6077	1·8945
6·9	21·677	37·3928	47·61	328·509	2·6268	1·9038
7·0	21·991	38·4845	49·00	343·000	2·6458	1·9129
7·1	22·305	39·5919	50·41	357·911	2·6646	1·9220
7·2	22·619	40·7150	51·84	373·248	2·6833	1·9310
7·3	22·934	41·8539	53·29	389·017	2·7019	1·9399
7·4	23·248	43·0084	54·76	405·224	2·7203	1·9487
7·5	23·562	44·1786	56·25	421·875	2·7386	1·9574
7·6	23·876	45·3646	57·76	438·976	2·7568	1·9661
7·7	24·190	46·5663	59·29	456·533	2·7749	1·9747
7·8	24·504	47·7836	60·84	474·552	2·7928	1·9832
7·9	24·819	49·0167	62·41	493·039	2·8107	1·9916
8·0	25·133	50·2655	64·00	512·000	2·8284	2·0000
8·1	25·447	51·5300	65·61	531·441	2·8461	2·0083
8·2	25·761	52·8102	67·24	551·368	2·8636	2·0165
8·3	26·075	54·1061	68·89	571·787	2·8810	2·0247
8·4	26·389	55·4177	70·56	592·704	2·8983	2·0328
8·5	26·704	56·7450	72·25	614·125	2·9155	2·0408
8·6	27·018	58·0880	73·96	636·056	2·9326	2·0488
8·7	27·332	59·4468	75·69	658·503	2·9496	2·0567
8·8	27·646	60·8212	77·44	681·472	2·9665	2·0646
8·9	27·960	62·2114	79·21	704·969	2·9833	2·0724
9·0	28·274	63·6173	81·00	729·000	3·0000	2·0801
9·1	28·588	65·0388	82·81	753·571	3·0166	2·0878
9·2	28·903	66·4761	84·64	778·688	3·0332	2·0954
9·3	29·217	67·9291	86·49	804·357	3·0496	2·1029
9·4	29·531	69·3978	88·36	830·584	3·0659	2·1105
9·5	29·845	70·8822	90·25	857·375	3·0822	2·1179
9·6	30·159	72·3823	92·16	884·736	3·0984	2·1253
9·7	30·473	73·8981	94·09	912·673	3·1145	2·1327
9·8	30·788	75·4296	96·04	941·192	3·1305	2·1400
9·9	31·102	76·9769	98·01	970·299	3·1464	2·1472
10·0	31·416	78·540	100·00	1000·000	3·1623	2·1544
10·1	31·730	80·119	102·01	1030·301	3·1780	2·1616
10·2	32·044	81·713	104·04	1061·208	3·1937	2·1687
10·3	32·358	83·323	106·09	1092·727	3·2094	2·1757

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
10·4	32·673	84·949	108·16	1124·863	3·2249	2·1828
10·5	32·987	86·590	110·25	1157·625	3·2404	2·1897
10·6	33·301	88·247	112·36	1191·016	3·2558	2·1967
10·7	33·615	89·920	114·49	1225·043	3·2711	2·2036
10·8	33·929	91·609	116·64	1259·712	3·2863	2·2104
10·9	34·243	93·313	118·81	1295·029	3·3015	2·2172
11·0	34·558	95·033	121·00	1331·000	3·3166	2·2239
11·1	34·872	96·769	123·21	1367·631	3·3317	2·2307
11·2	35·186	98·520	125·44	1404·928	3·3466	2·2374
11·3	35·500	100·29	127·69	1442·897	3·3615	2·2441
11·4	35·814	102·07	129·96	1481·544	3·3764	2·2506
11·5	36·128	103·87	132·25	1520·875	3·3912	2·2572
11·6	36·442	105·68	134·56	1560·896	3·4059	2·2637
11·7	36·757	107·51	136·89	1601·613	3·4205	2·2702
11·8	37·071	109·36	139·24	1643·032	3·4351	2·2766
11·9	37·385	111·22	141·61	1685·159	3·4496	2·2831
12·0	37·699	113·10	144·00	1728·000	3·4641	2·2894
12·1	38·013	114·99	146·41	1771·561	3·4785	2·2957
12·2	38·327	116·90	148·84	1815·848	3·4928	2·3021
12·3	38·642	118·82	151·29	1860·867	3·5071	2·3084
12·4	38·956	120·76	153·76	1906·624	3·5214	2·3146
12·5	39·270	122·72	156·25	1953·125	3·5355	2·3208
12·6	39·584	124·69	158·76	2000·376	3·5496	2·3270
12·7	39·898	126·68	161·29	2048·383	3·5637	2·3331
12·8	40·212	128·68	163·84	2097·152	3·5777	2·3392
12·9	40·527	130·70	166·41	2146·689	3·5917	2·3453
13·0	40·841	132·73	169·00	2197·000	3·6056	2·3513
13·1	41·155	134·78	171·61	2248·091	3·6194	2·3573
13·2	41·469	136·85	174·24	2299·968	3·6332	2·3633
13·3	41·783	138·93	176·89	2352·637	3·6469	2·3693
13·4	42·097	141·03	179·56	2406·104	3·6606	2·3752
13·5	42·412	143·14	182·25	2460·375	3·6742	2·3811
13·6	42·726	145·27	184·96	2515·456	3·6878	2·3870
13·7	43·040	147·41	187·69	2571·353	3·7013	2·3928
13·8	43·354	149·57	190·44	2628·072	3·7148	2·3986
13·9	43·668	151·75	193·21	2685·619	3·7283	2·4044
14·0	43·982	153·94	196·00	2744·000	3·7417	2·4101
14·1	44·296	156·15	198·81	2803·221	3·7550	2·4159
14·2	44·611	158·37	201·64	2863·288	3·7683	2·4216
14·3	44·925	160·61	204·49	2924·207	3·7815	2·4372
14·4	45·239	162·86	207·36	2985·984	3·7947	2·4329

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
14·5	45·553	165·13	210·25	3048·625	3·8079	2·4385
14·6	45·867	167·42	213·16	3112·136	3·8210	2·4441
14·7	46·181	169·72	216·09	3176·523	3·8341	2·4497
14·8	46·496	172·03	219·04	3241·792	3·8471	2·4552
14·9	46·810	174·37	222·01	3307·949	3·8600	2·4607
15·0	47·124	176·715	225·00	3375·000	3·8730	2·4662
15·1	47·438	179·079	228·01	3442·951	3·8859	2·4717
15·2	47·752	181·458	231·04	3511·808	3·8987	2·4771
15·3	48·066	183·854	234·09	3581·577	3·9115	2·4825
15·4	48·381	186·265	237·16	3652·264	3·9243	2·4879
15·5	48·695	188·692	240·25	3723·875	3·9370	2·4933
15·6	49·009	191·134	243·36	3796·416	3·9497	2·4987
15·7	49·323	193·593	246·49	3869·893	3·9623	2·5040
15·8	49·637	196·067	249·64	3944·312	3·9749	2·5093
15·9	49·951	198·557	252·81	4019·679	3·9875	2·5146
16·0	50·265	201·062	256·00	4096·000	4·0000	2·5198
16·1	50·580	203·583	259·21	4173·281	4·0125	2·5251
16·2	50·894	206·120	262·44	4251·528	4·0249	2·5303
16·3	51·208	208·627	265·69	4330·747	4·0373	2·5355
16·4	51·522	211·241	268·96	4410·944	4·0497	2·5407
16·5	51·836	213·825	272·25	4492·125	4·0620	2·5458
16·6	52·150	216·424	275·56	4574·296	4·0743	2·5509
16·7	52·465	219·040	278·89	4657·463	4·0866	2·5561
16·8	52·779	221·671	282·24	4741·632	4·0988	2·5612
16·9	53·093	224·318	285·61	4826·809	4·1110	2·5662
17·0	53·407	226·980	289·00	4913·000	4·1231	2·5713
17·1	53·721	229·658	292·41	5000·211	4·1352	2·5763
17·2	54·035	232·352	295·84	5088·448	4·1473	2·5813
17·3	54·350	235·062	299·29	5177·717	4·1593	2·5863
17·4	54·664	237·787	302·76	5268·024	4·1713	2·5913
17·5	54·978	240·528	306·25	5359·375	4·1833	2·5962
17·6	55·292	243·285	309·76	5451·776	4·1952	2·6012
17·7	55·606	246·057	313·29	5545·233	4·2071	2·6061
17·8	55·920	248·846	316·84	5639·752	4·2190	2·6110
17·9	56·235	251·649	320·41	5735·339	4·2308	2·6159
18·0	56·549	254·469	324·00	5832·000	4·2426	2·6207
18·1	56·863	257·304	327·61	5929·741	4·2544	2·6256
18·2	57·177	260·155	331·24	6028·568	4·2661	2·6304
18·3	57·491	263·022	334·89	6128·487	4·2778	2·6352
18·4	57·805	265·904	338·56	6229·504	4·2895	2·6400
18·5	58·119	268·803	342·25	6331·625	4·3012	2·6448

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2\frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
18·6	58·434	271·716	345·96	6434·856	4·3128	2·6495
18·7	58·748	274·646	349·69	6539·203	4·3243	2·6543
18·8	59·062	277·591	353·44	6644·672	4·3359	2·6590
18·9	59·376	280·552	357·21	6751·269	4·3474	2·6637
19·0	59·690	283·529	361·00	6859·000	4·3589	2·6684
19·1	60·004	286·521	364·81	6967·871	4·3704	2·6731
19·2	60·319	289·529	368·64	7077·888	4·3818	2·6777
19·3	60·633	292·553	372·49	7189·057	4·3932	2·6824
19·4	60·947	295·592	376·36	7301·384	4·4045	2·6870
19·5	61·261	298·648	380·25	7414·875	4·4159	2·6916
19·6	61·575	301·719	384·16	7529·536	4·4272	2·6962
19·7	61·889	304·805	388·09	7645·373	4·4385	2·7008
19·8	62·204	307·907	392·04	7762·392	4·4497	2·7053
19·9	62·518	311·026	396·01	7880·599	4·4609	2·7099
20·0	62·832	314·16	400·00	8000·000	4·4721	2·7144
20·1	63·146	317·31	404·01	8120·601	4·4833	2·7189
20·2	63·460	320·47	408·04	8242·408	4·4944	2·7234
20·3	63·774	323·66	412·09	8365·427	4·5055	2·7279
20·4	64·088	326·85	416·16	8489·664	4·5166	2·7324
20·5	64·403	330·06	420·25	8615·125	4·5277	2·7368
20·6	64·717	333·29	424·36	8741·816	4·5387	2·7413
20·7	65·031	336·54	428·49	8869·743	4·5497	2·7457
20·8	65·345	339·80	432·64	8998·912	4·5607	2·7502
20·9	65·659	343·07	436·81	9129·329	4·5716	2·7545
21·0	65·973	346·36	441·00	9261·000	4·5826	2·7589
21·1	66·288	349·67	445·21	9393·931	4·5935	2·7633
21·2	66·602	352·99	449·44	9528·128	4·6043	2·7676
21·3	66·916	356·33	453·69	9663·597	4·6152	2·7720
21·4	67·230	359·68	457·96	9800·344	4·6260	2·7763
21·5	67·544	363·05	462·25	9938·375	4·6368	2·7806
21·6	67·858	366·44	466·56	10077·696	4·6476	2·7849
21·7	68·173	369·84	470·89	10218·313	4·6583	2·7893
21·8	68·487	373·25	475·24	10360·232	4·6690	2·7935
21·9	68·801	376·69	479·61	10503·459	4·6797	2·7978
22·0	69·115	380·13	484·00	10648·000	4·6904	2·8021
22·1	69·429	383·60	488·41	10793·861	4·7011	2·8063
22·2	69·743	387·08	492·84	10941·048	4·7117	2·8105
22·3	70·058	390·57	497·29	11089·567	4·7223	2·8147
22·4	70·372	394·08	501·76	11239·424	4·7329	2·8189
22·5	70·686	397·61	506·25	11390·625	4·7434	2·8231
22·6	71·000	401·15	510·76	11543·176	4·7539	2·8273

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
22·7	71·314	404·71	515·29	11697·083	4·7644	2·8314
22·8	71·628	408·28	519·84	11852·352	4·7749	2·8356
22·9	71·942	411·87	524·41	12008·989	4·7854	2·8397
23·0	72·257	415·48	529·00	12167·000	4·7958	2·8438
23·1	72·571	419·10	533·61	12326·391	4·8062	2·8479
23·2	72·885	422·73	538·24	12487·168	4·8166	2·8521
23·3	73·199	426·39	542·89	12649·337	4·8270	2·8562
23·4	73·513	430·05	547·56	12812·904	4·8373	2·8603
23·5	73·827	433·74	552·25	12977·875	4·8477	2·8643
23·6	74·142	437·44	556·96	13144·256	4·8580	2·8684
23·7	74·456	441·15	561·69	13312·053	4·8683	2·8724
23·8	74·770	444·88	566·44	13481·272	4·8785	2·8765
23·9	75·084	448·63	571·21	13651·919	4·8888	2·8805
24·0	75·398	452·39	576·00	13824·000	4·8990	2·8845
24·1	75·712	456·17	580·81	13997·521	4·9092	2·8885
24·2	76·027	459·96	585·64	14172·488	4·9193	2·8925
24·3	76·341	463·77	590·49	14348·907	4·9295	2·8965
24·4	76·655	467·60	595·36	14526·784	4·9396	2·9004
24·5	76·969	471·44	600·25	14706·125	4·9497	2·9044
24·6	77·283	475·29	605·16	14886·936	4·9598	2·9083
24·7	77·597	479·16	610·09	15069·223	4·9699	2·9123
24·8	77·911	483·05	615·04	15252·992	4·9799	2·9162
24·9	78·226	486·96	610·01	15438·249	4·9899	2·9201
25·0	78·540	490·874	625·00	15625·000	5·0000	2·9240
25·1	78·854	494·809	630·01	15813·251	5·0099	2·9279
25·2	79·168	498·759	635·04	16003·008	5·0199	2·9318
25·3	79·482	502·726	640·09	16194·277	5·0299	2·9357
25·4	79·796	506·708	645·16	16387·064	5·0398	2·9395
25·5	80·111	510·705	650·25	16581·375	5·0498	2·9434
25·6	80·425	514·719	655·36	16777·216	5·0596	2·9472
25·7	80·739	518·748	660·49	16974·593	5·0695	2·9511
25·8	81·053	522·792	665·64	17173·512	5·0794	2·9549
25·9	81·367	526·853	670·81	17373·979	5·0892	2·9587
26·0	81·681	530·929	676·00	17576·000	5·0990	2·9625
26·1	81·996	535·021	681·21	17779·581	5·1088	2·9663
26·2	82·310	539·129	686·44	17984·728	5·1186	2·9701
26·3	82·624	543·252	691·69	18191·447	5·1284	2·9738
26·4	82·938	547·391	696·96	18399·744	5·1381	2·9776
26·5	83·252	551·546	702·25	18609·625	5·1478	2·9814
26·6	83·566	555·716	707·56	18821·096	5·1575	2·9851
26·7	83·881	559·903	712·89	19034·163	5·1672	2·9888

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2\frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
26·8	84·195	564·104	718·24	19248·832	5·1769	2·9926
26·9	84·509	568·322	723·61	19465·109	5·1865	2·9963
27·0	84·823	572·555	729·00	19683·000	5·1962	3·0000
27·1	85·137	576·804	734·41	19902·511	5·2058	3·0037
27·2	85·451	581·069	739·84	20123·648	5·2154	3·0074
27·3	85·765	585·349	745·29	20346·417	5·2249	3·0111
27·4	86·080	589·646	750·76	20570·824	5·2345	3·0147
27·5	86·394	593·957	756·25	20796·875	5·2440	3·0184
27·6	86·708	598·285	761·76	21024·576	5·2536	3·0221
27·7	87·022	602·628	767·29	21253·933	5·2631	3·0257
27·8	87·336	606·987	772·84	21484·952	5·2726	3·0293
27·9	87·650	611·362	778·41	21717·639	5·2820	3·0330
28·0	87·965	615·752	784·00	21952·000	5·2915	3·0366
28·1	88·279	620·158	789·61	22188·041	5·3009	3·0402
28·2	88·593	624·580	795·24	22425·768	5·3104	3·0438
28·3	88·907	629·018	800·89	22665·187	5·3198	3·0474
28·4	89·221	633·471	806·56	22906·304	5·3292	3·0510
28·5	89·535	637·940	812·25	23149·125	5·3385	3·0546
28·6	89·850	642·424	817·96	23393·656	5·3479	3·0581
28·7	90·164	646·925	823·69	23639·903	5·3572	3·0617
28·8	90·478	651·441	829·44	23887·872	5·3666	3·0652
28·9	90·792	655·972	835·21	24137·569	5·3759	3·0688
29·0	91·106	660·520	841·00	24389·000	5·3852	3·0723
29·1	91·420	665·083	846·81	24642·171	5·3944	3·0758
29·2	91·735	669·662	852·64	24897·088	5·4037	3·0794
29·3	92·049	674·256	858·49	25153·757	5·4129	3·0829
29·4	92·363	678·867	864·36	25412·184	5·4222	3·0864
29·5	92·677	683·493	870·25	25672·375	5·4314	3·0899
29·6	92·991	688·134	876·16	25934·336	5·4406	3·0934
29·7	93·305	692·792	882·09	26198·073	5·4498	3·0968
29·8	93·619	697·465	888·04	26463·592	5·4589	3·1003
29·9	93·934	702·154	894·01	26730·899	5·4681	3·1038
30·0	94·248	706·86	900·00	27000·000	5·4772	3·1072
30·1	94·562	711·58	906·01	27270·901	5·4863	3·1107
30·2	94·876	716·32	912·04	27543·608	5·4954	3·1141
30·3	95·190	721·07	918·09	27818·127	5·5045	3·1176
30·4	95·504	725·83	924·16	28094·464	5·5136	3·1210
30·5	95·819	730·62	930·25	28372·625	5·5226	3·1244
30·6	96·133	735·42	936·36	28652·616	5·5317	3·1278
30·7	96·447	740·23	942·49	28934·443	5·5407	3·1312
30·8	96·761	745·06	948·64	29218·112	5·5497	3·1346

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2\frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
30·9	97·075	749·91	954·81	29503·629	5·5587	3·1380
31·0	97·389	754·77	961·00	29791·000	5·5678	3·1414
31·1	97·704	759·65	967·21	30080·231	5·5767	3·1448
31·2	98·018	764·54	973·44	30371·328	5·5857	3·1481
31·3	98·332	769·45	979·69	30664·297	5·5946	3·1515
31·4	98·646	774·37	985·96	30959·144	5·6035	3·1548
31·5	98·960	779·31	992·25	31255·875	5·6124	3·1582
31·6	99·274	784·27	998·56	31554·496	5·6213	3·1615
31·7	99·588	789·24	1004·89	31855·013	5·6302	3·1648
31·8	99·903	794·23	1011·24	32157·432	5·6391	3·1681
31·9	100·22	799·23	1017·61	32461·759	5·6480	3·1715
32·0	100·53	804·25	1024·00	32768·000	5·6569	3·1748
32·1	100·85	809·28	1030·41	33076·161	5·6656	3·1781
32·2	101·16	814·33	1036·84	33386·248	5·6745	3·1814
32·3	101·47	819·40	1043·29	33698·267	5·6833	3·1847
32·4	101·79	824·48	1049·76	34012·224	5·6921	3·1880
32·5	102·10	829·58	1056·25	34328·125	5·7008	3·1913
32·6	102·42	834·69	1062·76	34645·976	5·7096	3·1945
32·7	102·73	839·82	1069·29	34965·783	5·7183	3·1978
32·8	103·04	844·96	1075·84	35287·552	5·7271	3·2010
32·9	103·36	850·12	1082·41	35611·289	5·7358	3·2043
33·0	103·67	855·30	1089·00	35937·000	5·7446	3·2075
33·1	103·99	860·49	1095·61	36264·691	5·7532	3·2108
33·2	104·30	865·70	1102·24	36594·368	5·7619	3·2140
33·3	104·62	870·92	1108·89	36926·037	5·7706	3·2172
33·4	104·93	876·16	1115·56	37259·704	5·7792	3·2204
33·5	105·24	881·41	1122·25	37595·375	5·7879	3·2237
33·6	105·56	886·68	1128·96	37933·056	5·7965	3·2269
33·7	105·87	891·97	1135·69	38272·753	5·8051	3·2301
33·8	106·19	897·27	1142·44	38614·472	5·8137	3·2332
33·9	106·50	902·59	1149·21	38958·219	5·8223	3·2364
34·0	106·81	907·92	1156·00	39304·000	5·8310	3·2396
34·1	107·13	913·27	1162·81	39651·821	5·8395	3·2428
34·2	107·44	918·63	1169·64	40001·688	5·8480	3·2460
34·3	107·76	924·01	1176·49	40353·607	5·8566	3·2491
34·4	108·07	929·41	1183·36	40707·584	5·8651	3·2522
34·5	108·38	934·82	1190·25	41063·625	5·8736	3·2554
34·6	108·70	940·25	1197·16	41421·736	5·8821	3·2586
34·7	109·01	945·69	1204·09	41781·923	5·8906	3·2617
34·8	109·33	951·15	1211·04	42144·192	5·8991	3·2648
34·9	109·64	956·62	1218·01	42508·549	5·9076	3·2679

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
35·0	109·96	962·113	1225·00	42875·000	5·9161	3·2711
35·1	110·27	967·618	1232·01	43243·551	5·9245	3·2742
35·2	110·58	973·140	1239·04	43614·208	5·9330	3·2773
35·3	110·90	978·677	1246·09	43986·977	5·9414	3·2804
35·4	111·21	984·230	1253·16	44361·864	5·9498	3·2835
35·5	111·53	989·798	1260·25	44738·875	5·9582	3·2866
35·6	111·84	995·382	1267·36	45118·016	5·9666	3·2897
35·7	112·15	1000·98	1274·49	45499·293	5·9749	3·2927
35·8	112·47	1006·60	1281·64	45882·712	5·9833	3·2958
35·9	112·78	1012·23	1288·81	46268·279	5·9917	3·2989
36·0	113·10	1017·88	1296·00	46656·000	6·0000	3·3019
36·1	113·41	1023·54	1303·21	47045·881	6·0083	3·3050
36·2	113·73	1029·22	1310·44	47437·928	6·0166	3·3080
36·3	114·04	1034·91	1317·69	47832·147	6·0249	3·3111
36·4	114·35	1040·62	1324·96	48228·544	6·0332	3·3141
36·5	114·67	1046·35	1332·25	48627·125	6·0415	3·3171
36·6	114·98	1052·09	1339·56	49027·896	6·0498	3·3202
36·7	115·30	1057·84	1346·89	49430·863	6·0581	3·3232
36·8	115·61	1063·62	1354·24	49836·032	6·0663	3·3262
36·9	115·92	1069·41	1361·61	50243·409	6·0745	3·3292
37·0	116·24	1075·21	1369·00	50653·000	6·0828	3·3322
37·1	116·55	1081·03	1376·41	51064·811	6·0910	3·3352
37·2	116·87	1086·87	1383·84	51478·848	6·0992	3·3382
37·3	117·18	1092·72	1391·29	51895·117	6·1074	3·3412
37·4	117·50	1098·58	1398·76	52313·624	6·1156	3·3442
37·5	117·81	1104·47	1406·25	52734·375	6·1237	3·3472
37·6	118·12	1110·36	1413·76	53157·376	6·1319	3·3501
37·7	118·44	1116·28	1421·29	53582·633	6·1400	3·3531
37·8	118·75	1122·21	1428·84	54010·152	6·1482	3·3561
37·9	119·07	1128·15	1436·41	54439·939	6·1563	3·3590
38·0	119·38	1134·11	1444·00	54872·000	6·1644	3·3620
38·1	119·69	1140·09	1451·61	55306·341	6·1725	3·3649
38·2	120·01	1146·08	1459·24	55742·968	6·1806	3·3679
38·3	120·32	1152·09	1466·89	56181·887	6·1887	3·3708
38·4	120·64	1158·12	1474·56	56623·104	6·1968	3·3737
38·5	120·95	1164·16	1482·25	57066·625	6·2048	3·3767
38·6	121·27	1170·21	1489·96	57512·456	6·2129	3·3796
38·7	121·58	1176·28	1497·69	57960·603	6·2209	3·3825
38·8	121·89	1182·37	1505·44	58411·072	6·2290	3·3854
38·9	122·21	1188·47	1513·21	58863·869	6·2370	3·3883
39·0	122·52	1194·59	1521·00	59319·000	6·2450	3·3912

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2\frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
39·1	122·84	1200·72	1528·81	59776·471	6·2530	3·3941
39·2	123·15	1206·87	1536·64	60236·288	6·2610	3·3970
39·3	123·46	1213·04	1544·49	60698·457	6·2690	3·3999
39·4	123·78	1219·22	1552·36	61162·984	6·2769	3·4028
39·5	124·09	1225·42	1560·25	61629·875	6·2849	3·4056
39·6	124·41	1231·63	1568·16	62099·136	6·2929	3·4085
39·7	124·72	1237·86	1576·09	62570·773	6·3008	3·4114
39·8	125·04	1244·10	1584·04	63044·792	6·3087	3·4142
39·9	125·35	1250·36	1592·01	63521·199	6·3166	3·4171
40·0	125·66	1256·64	1600·00	64000·000	6·3246	3·4200
40·1	125·98	1262·93	1608·01	64481·201	6·3325	3·4228
40·2	126·29	1269·23	1616·04	64964·808	6·3404	3·4256
40·3	126·61	1275·56	1624·09	65450·827	6·3482	3·4285
40·4	126·92	1281·90	1632·16	65939·264	6·3561	3·4313
40·5	127·23	1288·25	1640·25	66430·125	6·3639	3·4341
40·6	127·55	1294·62	1648·36	66923·416	6·3718	3·4370
40·7	127·86	1301·00	1656·49	67419·143	6·3796	3·4398
40·8	128·18	1307·41	1664·64	67911·312	6·3875	3·4426
40·9	128·49	1313·82	1672·81	68417·929	6·3953	3·4454
41·0	128·81	1320·25	1681·00	68921·000	6·4031	3·4482
41·1	129·12	1326·70	1689·21	69426·531	6·4109	3·4510
41·2	129·43	1333·17	1697·44	69934·528	6·4187	3·4538
41·3	129·75	1339·65	1705·69	70444·997	6·4265	3·4566
41·4	130·06	1346·14	1713·96	70957·944	6·4343	3·4594
41·5	130·38	1352·65	1722·25	71473·375	6·4421	3·4622
41·6	130·69	1359·18	1730·56	71991·296	6·4498	3·4650
41·7	131·00	1365·72	1738·89	72511·713	6·4575	3·4677
41·8	131·32	1372·28	1747·24	73034·632	6·4653	3·4705
41·9	131·63	1378·85	1755·61	73560·059	6·4730	3·4733
42·0	131·95	1385·44	1764·00	74088·000	6·4807	3·4760
42·1	132·26	1392·05	1772·41	74618·461	6·4884	3·4788
42·2	132·58	1398·67	1780·84	75151·448	6·4961	3·4815
42·3	132·89	1405·31	1789·29	75686·967	6·5038	3·4843
42·4	133·20	1411·96	1797·76	76225·024	6·5115	3·4870
42·5	133·52	1418·63	1806·25	76765·625	6·5192	3·4898
42·6	133·83	1425·31	1814·76	77308·776	6·5268	3·4925
42·7	134·15	1432·01	1823·29	77854·483	6·5345	3·4952
42·8	134·46	1438·72	1831·84	78402·752	6·5422	3·4980
42·9	134·77	1445·45	1840·41	78953·589	6·5498	3·5007
43·0	135·09	1452·20	1849·00	79507·000	6·5574	3·5034
43·1	135·40	1458·96	1857·61	80062·991	6·5651	3·5061

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
43·2	135·72	1465·74	1866·24	80621·568	6·5727	3·5088
43·3	136·03	1472·54	1874·89	81182·737	6·5803	3·5115
43·4	136·35	1479·34	1883·56	81746·504	6·5879	3·5142
43·5	136·66	1486·17	1892·25	82312·875	6·5954	3·5169
43·6	136·97	1493·01	1900·96	82881·856	6·6030	3·5196
43·7	137·29	1499·87	1909·69	83453·453	6·6106	3·5223
43·8	137·60	1506·74	1918·44	84027·672	6·6182	3·5250
43·9	137·92	1513·63	1927·21	84604·519	6·6257	3·5277
44·0	138·23	1520·53	1936·00	85184·000	6·6332	3·5303
44·1	138·54	1527·45	1944·81	85766·121	6·6408	3·5330
44·2	138·86	1534·39	1953·64	86350·888	6·6483	3·5357
44·3	139·17	1541·34	1962·49	86938·307	6·6558	3·5384
44·4	139·49	1548·30	1971·36	87528·384	6·6633	3·5410
44·5	139·80	1555·28	1980·25	88121·125	6·6708	3·5437
44·6	140·12	1562·28	1989·16	88716·536	6·6783	3·5463
44·7	140·43	1569·30	1998·09	89314·623	6·6858	3·5490
44·8	140·74	1576·33	2007·04	89915·392	6·6933	3·5516
44·9	141·06	1583·37	2016·01	90518·849	6·7007	3·5543
45·0	141·37	1590·43	2025·00	91125·000	6·7082	3·5569
45·1	141·69	1597·51	2034·01	91733·851	6·7157	3·5595
45·2	142·00	1604·60	2043·04	92345·408	6·7231	3·5622
45·3	142·31	1611·71	2052·09	92959·677	6·7305	3·5648
45·4	142·63	1618·83	2061·16	93576·664	6·7380	3·5674
45·5	142·94	1625·97	2070·25	94196·375	6·7454	3·5700
45·6	143·26	1633·13	2079·36	94818·816	6·7528	3·5726
45·7	143·57	1640·30	2088·49	95443·993	6·7602	3·5752
45·8	143·88	1647·48	2097·64	96071·912	6·7676	3·5778
45·9	144·20	1654·68	2106·81	96702·579	6·7750	3·5805
46·0	144·51	1661·90	2116·00	97336·000	6·7823	3·5830
46·1	144·83	1669·14	2125·21	97972·181	6·7897	3·5856
46·2	145·14	1676·39	2134·44	98611·128	6·7971	3·5882
46·3	145·46	1683·65	2143·69	99252·847	6·8044	3·5908
46·4	145·77	1690·93	2152·96	99897·344	6·8118	3·5934
46·5	146·08	1698·23	2162·25	100544·625	6·8191	3·5960
46·6	146·40	1705·54	2171·56	101194·696	6·8264	3·5986
46·7	146·71	1712·87	2180·89	101847·563	6·8337	3·6011
46·8	147·03	1720·21	2190·24	102503·232	6·8411	3·6037
46·9	147·34	1727·57	2199·61	103161·709	6·8484	3·6063
47·0	147·65	1734·94	2209·00	103823·000	6·8557	3·6088
47·1	147·97	1742·34	2218·41	104487·111	6·8629	3·6114
47·2	148·28	1749·74	2227·84	105154·048	6·8702	3·6139

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
47·3	148·60	1757·16	2237·29	105823·817	6·8775	3·6165
47·4	148·91	1764·60	2246·76	106496·424	6·8848	3·6190
47·5	149·23	1772·05	2256·25	107171·875	6·8920	3·6216
47·6	149·54	1779·52	2265·76	107850·176	6·8993	3·6241
47·7	149·85	1787·01	2275·29	108531·333	6·9065	3·6267
47·8	150·17	1794·51	2284·84	109215·352	6·9138	3·6292
47·9	150·48	1802·03	2294·41	109902·239	6·9210	3·6317
48·0	150·80	1809·56	2304·00	110592·000	6·9282	3·6342
48·1	151·11	1817·11	2313·61	111284·641	6·9354	3·6368
48·2	151·42	1824·67	2323·24	111980·168	6·9426	3·6393
48·3	151·74	1832·25	2332·89	112678·587	6·9498	3·6418
48·4	152·05	1839·84	2342·56	113379·904	6·9570	3·6443
48·5	152·37	1847·45	2352·25	114084·125	6·9642	3·6468
48·6	152·68	1855·08	2361·96	114791·256	6·9714	3·6493
48·7	153·00	1862·72	2371·69	115501·303	6·9785	3·6518
48·8	153·31	1870·38	2381·44	116214·272	6·9857	3·6543
48·9	153·62	1878·05	2391·21	116930·169	6·9929	3·6568
49·0	153·94	1885·74	2401·00	117649·000	7·0000	3·6593
49·1	154·25	1893·45	2410·81	118370·771	7·0071	3·6618
49·2	154·57	1901·17	2420·64	119095·488	7·0143	3·6643
49·3	154·88	1908·90	2430·49	119823·157	7·0214	3·6668
49·4	155·19	1916·65	2440·36	120553·784	7·0285	3·6692
49·5	155·51	1924·42	2450·25	121287·375	7·0356	3·6717
49·6	155·82	1932·21	2460·16	122023·936	7·0427	3·6742
49·7	156·14	1940·00	2470·09	122763·473	7·0498	3·6766
49·8	156·45	1947·82	2480·04	123505·992	7·0569	3·6791
49·9	156·77	1955·65	2490·01	124251·499	7·0640	3·6816
50·0	157·08	1963·50	2500·00	125000·000	7·0711	3·6840
51·0	160·22	2042·82	2601·00	132651·000	7·1414	3·7084
52·0	163·36	2123·72	2704·00	140608·000	7·2111	3·7325
53·0	166·50	2206·19	2809·00	148877·000	7·2801	3·7563
54·0	169·64	2290·22	2916·00	157464·000	7·3485	3·7798
55·0	172·78	2375·83	3025·00	166375·000	7·4162	3·8030
56·0	175·93	2463·01	3136·00	175616·000	7·4833	3·8259
57·0	179·07	2551·76	3249·00	185193·000	7·5498	3·8485
58·0	182·21	2642·08	3364·00	195112·000	7·6158	3·8709
59·0	185·35	2733·97	3481·00	205379·000	7·6811	3·8930
60·0	188·49	2827·44	3600·00	216000·000	7·7460	3·9149
61·0	191·64	2922·47	3721·00	226981·000	7·8102	3·9365
62·0	194·77	3019·07	3844·00	238328·000	7·8740	3·9579
63·0	197·92	3117·25	3969·00	250047·000	7·9373	3·9791

TABLE OF POWERS (*continued*)

n	$n\pi$	$n^2 \frac{\pi}{4}$	n^2	n^3	$\sqrt{n}$	$\sqrt[3]{n}$
64·0	201·06	3216·99	4096·00	262144·000	8·0000	4·0000
65·0	204·20	3318·31	4225·00	274625·000	8·0623	4·0207
66·0	207·34	3421·20	4356·00	287496·000	8·1240	4·0412
67·0	210·49	3525·65	4489·00	300763·000	8·1854	4·0615
68·0	213·63	3631·68	4624·00	314432·000	8·2462	4·0817
69·0	216·77	3739·28	4761·00	328509·000	8·3066	4·1016
70·0	219·91	3848·45	4900·00	343000·000	8·3666	4·1213
71·0	223·05	3959·19	5041·00	357911·000	8·4261	4·1408
72·0	226·19	4071·50	5184·00	373248·000	8·4853	4·1602
73·0	229·34	4185·39	5329·00	389017·000	8·5440	4·1793
74·0	232·48	4300·84	5476·00	405224·000	8·6023	4·1983
75·0	235·62	4417·86	5625·00	421875·000	8·6603	4·2172
76·0	238·76	4536·46	5776·00	438976·000	8·7178	4·2358
77·0	241·90	4656·63	5929·00	456533·000	8·7750	4·2543
78·0	245·04	4778·36	6084·00	474552·000	8·8318	4·2727
79·0	248·19	4901·67	6241·00	493039·000	8·8882	4·2908
80·0	251·32	5026·55	6400·00	512000·000	8·9443	4·3089
81·0	254·47	5153·00	6561·00	531441·000	9·0000	4·3267
82·0	257·61	5281·02	6724·00	551368·000	9·0554	4·3445
83·0	260·75	5410·61	6889·00	571787·000	9·1104	4·3621
84·0	263·89	5541·77	7056·00	592704·000	9·1652	4·3795
85·0	267·04	5674·50	7225·00	614125·000	9·2195	4·3968
86·0	270·18	5808·80	7396·00	636056·000	9·2736	4·4140
87·0	273·32	5944·68	7569·00	658503·000	9·3274	4·4310
88·0	276·46	6082·12	7744·00	681472·000	9·3808	4·4480
89·0	279·60	6221·14	7921·00	704969·000	9·4340	4·4647
90·0	282·74	6361·73	8100·00	729000·000	9·4868	4·4814
91·0	285·88	6503·88	8281·00	753571·000	9·5394	4·4979
92·0	289·03	6647·61	8464·00	778688·000	9·5917	4·5144
93·0	292·17	6792·91	8649·00	804357·000	9·6437	4·5307
94·0	295·31	6939·78	8836·00	830584·000	9·6954	4·5468
95·0	298·45	7088·22	9025·00	857375·000	9·7468	4·5629
96·0	301·59	7238·23	9216·00	884736·000	9·7980	4·5789
97·0	304·73	7389·81	9409·00	912673·000	9·8489	4·5947
98·0	307·88	7542·96	9604·00	941192·000	9·8995	4·6104
99·0	311·02	7697·69	9801·00	970299·000	9·9499	4·6261
100·0	314·16	7853·98	10000·00	1000000·000	10·0000	4·6416

TABLE OF LENGTHS OF CIRCULAR ARCS, CHORDS, AND HEIGHTS OF ARCS TO RADIUS 1

Degrees	Length of Arc	Chord	Height of Arc	Degrees	Length of Arc	Chord	Height of Arc
1	0·0175	0·0175	0·0000	36	0·6283	0·6180	0·0489
2	0·0349	0·0349	0·0002	37	0·6458	0·6346	0·0517
3	0·0524	0·0524	0·0003	38	0·6632	0·6511	0·0545
4	0·0698	0·0698	0·0006	39	0·6807	0·6676	0·0574
5	0·0873	0·0872	0·0010	40	0·6981	0·6840	0·0603
6	0·1047	0·1047	0·0014	41	0·7156	0·7004	0·0633
7	0·1222	0·1221	0·0019	42	0·7330	0·7167	0·0664
8	0·1396	0·1395	0·0024	43	0·7505	0·7330	0·0696
9	0·1571	0·1569	0·0031	44	0·7679	0·7492	0·0728
10	0·1745	0·1743	0·0038	45	0·7854	0·7654	0·0761
11	0·1920	0·1917	0·0046	46	0·8029	0·7815	0·0795
12	0·2094	0·2091	0·0055	47	0·8203	0·7975	0·0829
13	0·2269	0·2264	0·0064	48	0·8378	0·8135	0·0865
14	0·2443	0·2437	0·0075	49	0·8552	0·8294	0·0900
15	0·2618	0·2611	0·0086	50	0·8727	0·8452	0·0937
16	0·2793	0·2783	0·0097	51	0·8901	0·8610	0·0974
17	0·2967	0·2956	0·0110	52	0·9076	0·8767	0·1012
18	0·3142	0·3129	0·0123	53	0·9250	0·8924	0·1051
19	0·3316	0·3301	0·0137	54	0·9425	0·9080	0·1090
20	0·3491	0·3473	0·0152	55	0·9599	0·9235	0·1130
21	0·3665	0·3645	0·0167	56	0·9774	0·9389	0·1171
22	0·3840	0·3816	0·0184	57	0·9948	0·9543	0·1212
23	0·4014	0·3987	0·0201	58	1·0123	0·9696	0·1254
24	0·4189	0·4158	0·0219	59	1·0297	0·9848	0·1296
25	0·4363	0·4329	0·0237	60	1·0472	1·0000	0·1340
26	0·4538	0·4499	0·0256	61	1·0647	1·0151	0·1384
27	0·4712	0·4669	0·0276	62	1·0821	1·0301	0·1428
28	0·4887	0·4838	0·0297	63	1·0996	1·0450	0·1474
29	0·5061	0·5008	0·0319	64	1·1170	1·0598	0·1520
30	0·5236	0·5176	0·0341	65	1·1345	1·0746	0·1566
31	0·5411	0·5345	0·0364	66	1·1519	1·0893	0·1613
32	0·5585	0·5513	0·0387	67	1·1694	1·1039	0·1661
33	0·5760	0·5680	0·0412	68	1·1868	1·1184	0·1710
34	0·5934	0·5847	0·0437	69	1·2043	1·1328	0·1759
35	0·6109	0·6014	0·0463	70	1·2217	1·1472	0·1808

TABLE OF LENGTHS OF CIRCULAR ARCS, CHORDS, AND HEIGHTS OF ARCS TO RADIUS 1 (*continued*)

Degrees	Length of Arc	Chord	Height of Arc	Degrees	Length of Arc	Chord	Height of Arc
71	1·2392	1·1614	0·1859	106	1·8500	1·5972	0·3982
72	1·2566	1·1756	0·1910	107	1·8675	1·6077	0·4052
73	1·2741	1·1896	0·1961	108	1·8850	1·6180	0·4122
74	1·2915	1·2036	0·2014	109	1·9024	1·6282	0·4193
75	1·3090	1·2175	0·2066	110	1·9198	1·6383	0·4264
76	1·3265	1·2313	0·2120	111	1·9373	1·6483	0·4336
77	1·3439	1·2450	0·2174	112	1·9548	1·6581	0·4408
78	1·3614	1·2586	0·2229	113	1·9722	1·6678	0·4481
79	1·3788	1·2722	0·2284	114	1·9897	1·6773	0·4554
80	1·3963	1·2856	0·2340	115	2·0071	1·6868	0·4627
81	1·4137	1·2989	0·2396	116	2·0246	1·6961	0·4701
82	1·4312	1·3121	0·2453	117	2·0420	1·7053	0·4775
83	1·4486	1·3252	0·2510	118	2·0595	1·7143	0·4850
84	1·4661	1·3383	0·2569	119	2·0769	1·7233	0·4925
85	1·4835	1·3512	0·2627	120	2·0944	1·7321	0·5000
86	1·5010	1·3640	0·2686	121	2·1118	1·7407	0·5076
87	1·5184	1·3767	0·2746	122	2·1293	1·7492	0·5152
88	1·5359	1·3893	0·2807	123	2·1468	1·7576	0·5228
89	1·5533	1·4018	0·2867	124	2·1642	1·7659	0·5305
90	1·5708	1·4142	0·2929	125	2·1817	1·7740	0·5383
91	1·5882	1·4265	0·2991	126	2·1991	1·7820	0·5460
92	1·6057	1·4387	0·3053	127	2·2166	1·7899	0·5538
93	1·6232	1·4507	0·3116	128	2·2340	1·7976	0·5616
94	1·6406	1·4627	0·3180	129	2·2545	1·8052	0·5695
95	1·6580	1·4746	0·3244	130	2·2689	1·8126	0·5774
96	1·6755	1·4863	0·3309	131	2·2864	1·8199	0·5853
97	1·6930	1·4979	0·3374	132	2·3038	1·8271	0·5933
98	1·7104	1·5094	0·3439	133	2·3213	1·8341	0·6013
99	1·7279	1·5208	0·3506	134	2·3387	1·8410	0·6093
100	1·7453	1·5321	0·3572	135	2·3562	1·8478	0·6173
101	1·7628	1·5432	0·3639	136	2·3736	1·8544	0·6254
102	1·7802	1·5543	0·3707	137	2·3911	1·8608	0·6335
103	1·7977	1·5652	0·3775	138	2·4086	1·8672	0·6416
104	1·8151	1·5760	0·3843	139	2·4260	1·8733	0·6498
105	1·8326	1·5867	0·3912	140	2·4435	1·8794	0·6580

TABLE OF LENGTHS OF CIRCULAR ARCS, CHORDS, AND HEIGHTS OF ARCS TO RADIUS 1 (*continued*)

Degrees	Length of Arc	Chord	Height of Arc	Degrees	Length of Arc	Chord	Height of Arc
141	2·4609	1·8853	0·6662	161	2·8100	1·9726	0·8350
142	2·4784	1·8910	0·6744	162	2·8274	1·9754	0·8436
143	2·4958	1·8966	0·6827	163	2·8449	1·9780	0·8522
144	2·5133	1·9021	0·6910	164	2·8623	1·9805	0·8608
145	2·5307	1·9074	0·6993	165	2·8798	1·9829	0·8695
146	2·5482	1·9126	0·7076	166	2·8972	1·9851	0·8781
147	2·5656	1·9176	0·7160	167	2·9147	1·9871	0·8868
148	2·5831	1·9225	0·7244	168	2·9322	1·9890	0·8955
149	2·6005	1·9273	0·7328	169	2·9496	1·9908	0·9042
150	2·6180	1·9319	0·7412	170	2·9671	1·9924	0·9128
151	2·6354	1·9363	0·7496	171	2·9845	1·9938	0·9215
152	2·6529	1·9406	0·7581	172	3·0020	1·9951	0·9302
153	2·6704	1·9447	0·7666	173	3·0194	1·9963	0·9390
154	2·6878	1·9487	0·7750	174	3·0369	1·9973	0·9477
155	2·7053	1·9526	0·7836	175	3·0543	1·9981	0·9564
156	2·7227	1·9563	0·7921	176	3·0718	1·9988	0·9651
157	2·7402	1·9598	0·8006	177	3·0892	1·9993	0·9738
158	2·7576	1·9632	0·8092	178	3·1067	1·9997	0·9825
159	2·7751	1·9665	0·8178	179	3·1241	1·9999	0·9913
160	2·7925	1·9696	0·8264	180	3·1416	2·0000	1·0000

EXAMPLE. If we have an arc "*b d c*" of a circle whose radii *a b* or *a c* = 1, subtending an angle of 60°, the

Length of arc *b d c* . = 1·0472
Chord *b c* . . = 1·0000
Height *f d* of arc . = 0·1340

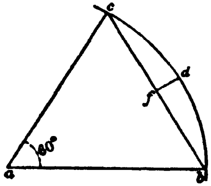


FIG. 4. ARC OF A CIRCLE

Useful Constants

1 gallon = 0·1604 cu ft = 10 lb of water at 63° F.

1 lb avoirdupois = 7000 gr = 453·6 g.

1 cu ft water weighs 62·3 lb.

1 cu ft air at 0° C weighs 0·0807 lb.

1 horse-power-hour = 33,000 × 60 foot-pounds.

1 electrical unit = 1000 watt-hours = 1·34 horse-power-hours.

1 horse-power = 33,000 ft-lb/min = 746 watts.

Volts × amperes = watts.

1 atmosphere = 14·7 lb/sq in. = 2116 lb/sq ft = 760 mm mercury.

A column of water 2·3 ft high corresponds to a pressure of 1 lb/sq in.

1 radian = 57·30 degrees.

$\pi = 3{\cdot}1416$.

1 micron = the thousandth part of a millimetre.

Mathematical Signs and Symbols

$+$	addition, plus, positive
$-$	subtraction, minus, negative
$\pm$ ($\mp$)	plus or minus (minus or plus)
$\times$ (.)	multiplication, by, times
$\div$	division, divided by
a/b	*a* divided by *b*
$\sim$	the difference between
$=$	equal to
$\bumpeq$	approximately equal to
$\neq$	not equal to
$>$	greater than
$<$	less than
$\propto$	varies as
∞	infinity
$\ngtr$	not greater than
$\nless$	not less than
$\geq$	greater than equal
$\leq$	less than equal
$:$	divided by, ratio sign, is to

Mathematical Signs and Symbols (*continued*)

$::$	proportional sign, as
$\because$	since, because
$\therefore$	therefore
$\sqrt[2]{}$	square root
$\sqrt[3]{}$	cube root
$\sqrt[n]{}$	*n*th root
a^n	*a* to the power of *n*
() [ ]	brackets
{ }	braces
—	vinculum or ( ) parenthesis, placed over two or more members of a compound quantity
$\angle$	angle
∟	right-angle
$\perp$	perpendicular
$\triangle$	triangle
□	square
▭	rectangle
○	circle
⌒	arc
$\parallel$	parallel
π	pi (3·14159)
°	degree
′	minutes, feet
″	seconds, inches
f	function

$\int$	integral of
$\int_B^A$	integral from *A* to *B*
d	differential
δ	variation
Σ	summation, sigma
M	the modulus of a system of logarithms
g	force of gravity
dx	differential of x
du/dx	partial derivative of u with respect to x
Δ	delta, finite, difference
Δx	increment of x
$4\vert$ $\lfloor\underline{4}$	factorial 1 × 2 × 3 × 4 $\lfloor\underline{4}$
$\vert x\vert$	absolute value of x irrespective of the sign
Å	Ångström
μ	micron 0·001 mm
log	logarithm
sin	sine
cos	cosine
tan	tangent
cot	cotangent
sec	secant
cosec	cosecant

HEAT

Thermometers. Comparison of Celsius, Réaumur, and Fahrenheit Scales

Celsius	Réaumur	Fahrenheit	Celsius	Réaumur	Fahrenheit
+ 100	+ 80	+ 212	+ 63	+ 50·4	+ 145·4
99	79·2	210·2	62	49·6	143·6
98	78·4	208·4	61	48·8	141·8
97	77·6	206·6	60	48	140
96	76·8	204·8	59	47·2	138·2
95	76	203	58	46·4	136·4
94	75·2	201·2	57	45·6	134·6
93	74·4	199·4	56	44·8	132·8
92	73·6	197·6	55	44	131
91	72·8	195·8	54	43·2	129·2
90	72	194	53	42·4	127·4
89	71·2	192·2	52	41·6	125·6
88	70·4	190·4	51	40·8	123·8
87	69·6	188·6	50	40	122
86	68·8	186·8	49	39·2	120·2
85	68	185	48	38·4	118·4
84	67·2	183·2	47	37·6	116·6
83	66·4	181·4	46	36·8	114·8
82	65·6	179·6	45	36	113
81	64·8	177·8	44	35·2	111·2
80	64	176	43	34·4	109·4
79	63·2	174·2	42	33·6	107·6
78	62·4	172·4	41	32·8	105·8
77	61·6	170·6	40	32	104
76	60·8	168·8	39	31·2	102·2
75	60	167	38	30·4	100·4
74	59·2	165·2	37	29·6	98·6
73	58·4	163·4	36	28·8	96·8
72	57·6	161·6	35	28	95
71	56·8	159·8	34	27·2	93·2
70	56	158	33	26·4	91·4
69	55·2	156·2	32	25·6	89·6
68	54·4	154·4	31	24·8	87·8
67	53·6	152·6	30	24	86
66	52·8	150·8	29	23·2	84·2
65	52	149	28	22·4	82·4
64	51·2	147·2	27	21·6	80·6

THERMOMETERS, COMPARISON OF (*continued*)

Celsius	Réaumur	Fahrenheit	Celsius	Réaumur	Fahrenheit
+ 26	+ 20·8	+ 78·8	− 8	− 6·4	+ 17·6
25	20	77	9	7·2	15·8
24	19·2	75·2	10	8	14
23	18·4	73·4	11	8·8	12·2
22	17·6	71·6	12	9·6	10·4
21	16·8	69·8	13	10·4	8·6
20	16	68	14	11·2	6·8
19	15·2	66·2	15	12	5
18	14·4	64·4	16	12·8	3·2
17	13·6	62·6	17	13·6	1·4
16	12·8	60·8	18	14·4	− 0·4
15	12	59	19	15·2	2·2
14	11·2	57·2	20	16	4
13	10·4	55·4	21	16·8	5·8
12	9·6	53·6	22	17·6	7·6
11	8·8	51·8	23	18·4	9·4
10	8	50	24	19·2	11·2
9	7·2	48·2	25	20	13
8	6·4	46·4	26	20·8	14·8
7	5·6	44·6	27	21·6	16·6
6	4·8	42·8	28	22·4	18·4
5	4	42	29	23·2	20·2
4	3·2	39·2	30	24	22
3	2·4	37·4	31	24·8	23·8
2	1·6	35·6	32	25·6	25·6
1	0·8	33·8	33	26·4	27·4
0	0	32	34	27·2	29·2
− 1	− 0·8	30·2	35	28	31
2	1·6	28·4	36	28·8	32·8
3	2·4	26·6	37	29·6	34·6
4	3·2	24·8	38	30·4	36·4
5	4	23	39	31·2	38·2
6	4·8	21·2	40	32	40
7	5·6	19·4			

$$t^\circ\ \text{C.} = \frac{4}{5}\,t^\circ\ \text{R.} = \frac{9}{5}\,.\,t^\circ + 32^\circ\ \text{F.}$$

$$t^\circ\ \text{R.} = \frac{5}{4}\,t^\circ\ \text{C.} = \frac{9}{4}\,.\,t^\circ + 32^\circ\ \text{F.}$$

$$t^\circ\ \text{F.} = \frac{5}{2}\,(t - 32)^\circ\ \text{C.} = \frac{4}{9}\,(t - 32)^\circ\ \text{R.}$$

TABLE OF APPROXIMATE TEMPERATURES INDICATED BY COLOUR OF FIRE

Glowing in the dark	525° C	Dark orange	1100° C
Dark red	700° C	Light orange	1200° C
Dark cherry red	800° C	White heat	1300° C
Cherry red	900° C	Welding heat	1400° C
Light cherry red	1000° C	Dazzling white	1500° C

Walker's List of Frigorific Mixtures

		Thermometer Sinks Degrees F.
Ammonium nitrate	1 part	From + 40° to + 4°
Water	1 ,,	
Ammonium chloride	5 parts	From + 50° to + 10°
Potassium nitrate	5 ,,	
Water	16 ,,	
Ammonium chloride	5 parts	From + 50° to + 4°
Potassium nitrate	5 ,,	
Sodium sulphate	8 ,,	
Water	16 ,,	
Sodium nitrate	3 parts	From + 50° to − 3°
Nitric acid, diluted	2 ,,	
Ammonic nitrate	1 part	From + 50° to − 7°
Sodium carbonate	1 ,,	
Water	1 ,,	
Sodium phosphate	9 parts	From + 50° to + 12°
Nitric acid, diluted	4 ,,	
Sodium sulphate	5 parts	From + 50° to + 3°
Sulphuric acid, diluted	4 ,,	
Sodium sulphate	6 parts	From + 50° to − 10°
Ammonium chloride	4 ,,	
Potassium nitrate	2 ,,	
Nitric acid, diluted	4 ,,	
Sodium sulphate	6 parts	From + 50° to − 40°
Ammonium nitrate	5 ,,	
Nitric acid, diluted	4 ,,	
Snow, or pounded ice	2 parts	to − 5°
Sodium chloride	1 part	
Snow, or pounded ice	5 parts	to − 12°
Sodium chloride	2 ,,	
Ammonium chloride	1 part	

		Thermometer Sinks Degrees F.
Snow, or pounded ice	24 parts	to − 18°
Sodium chloride	10 ,,	
Ammonium chloride	5 ,,	
Potassium nitrate	5 ,,	
Snow, or pounded ice	12 parts	to − 25°
Sodium chloride	5 ,,	
Ammonium nitrate	5 ,,	
Snow	3 parts	From + 32° to − 23°
Sulphuric acid, diluted	2 ,,	
Snow	8 parts	From + 32° to − 27°
Hydrochloric acid	5 ,,	
Snow	7 parts	From + 32° to − 30°
Nitric acid, diluted	4 ,,	
Snow	4 parts	From + 32° to − 40°
Calcium chloride	5 ,,	
Snow	2 parts	From + 32° to − 50°
Calcium chloride, crystallized	3 ,,	
Snow	3 parts	From + 32° to − 51°
Potash	4 ,,	

Heat-retaining Solution

A heat-retaining solution consists of a saturated solution of acetate of soda in hot water which will retain its heat for a long time, or until the acetate of soda crystallizes out. It can be used over and over again indefinitely by reheating till the crystals are again dissolved.

TRANSFORMATION OF COLUMNS OF WATER INTO COLUMNS OF MERCURY

Mm of Water	Mm of Mercury	Mm of Water	Mm of Mercury	Mm of Water	Mm of Mercury	Mm of Water	Mm of Mercury
1	0·074	8	0·59	35	2·58	65	4·80
2	0·15	9	0·66	40	2·95	70	5·17
3	0·22	10	0·74	45	3·32	75	5·54
4	0·30	15	1·12	50	3·69	80	5·90
5	0·37	20	1·48	55	4·06	85	6·27
6	0·44	25	1·84	60	4·43	90	6·64
7	0·52	30	2·21				

ELECTRICITY

Electric Units

Unit	Symbol	Name	Relation to other Units	C.G.S.
Current	C	Ampere	Volt ÷ ohm	10^{-1} or $\frac{1}{10}$
Quantity	Q	Coulomb	Ampere × second	10^{-1}
E.m.f.	E	Volt	Ampere × ohm	10^{8} or hundred millions
Resistance	R	Ohm	Volt ÷ ampere	10^{9} or thousand millions
Capacity	K	Farad	Coulomb ÷ volt	10^{-9}
Power	HP	Watt	Volt × ampere	10^{7}
Energy	W	Joule	Volt × coulomb	10^{7}

E.m.f. = electromotive force.

C.G.S. system is made up of the fundamental units: centimetre for length, the gramme for mass, and the second for time.

The unit for work is the watt-hour or the total work done when one watt is expended for one hour. A kilowatt-hour is the total work done when one kilowatt is expended for one hour. A watt is the rate of work represented by a current of one ampere urged by one volt. For convenience of speaking, and writing, of very large or small quantities of these units, certain prefixes are used—

$$\text{Meg-} = 1{,}000{,}000 \qquad \text{Micro-} = \frac{1}{1{,}000{,}000} \qquad \text{Kilo-} = 1000$$

$$\text{Milli-} = \frac{1}{1000} \qquad \text{Centi-} = \frac{1}{100} \qquad \text{Deci-} = \frac{1}{10}$$

For the purpose of briefly expressing large or small numbers, index notation is very useful. The method consists of writing the number as a power of 10 multiplied by a small number.

EXAMPLE. $100 = 10 \times 10 = 10^2$. $1000 = 10 \times 10 \times 10 = 10^3$.

Fractions may be denoted in a similar way—

$$\frac{1}{100} = 0{\cdot}01 = 10^{-2}. \quad \frac{1}{1000} = 0{\cdot}001 = 10^{-3}$$

A negative index indicates the fractional character of the number. The negative indices give the number of figures in the decimal form of the number, or the number of figures after the decimal point. In the large number, the index indicates the number of cyphers after the first or left-hand figure.

Some Electrical Terms

Accumulator or Condenser. A storage battery.

Ampere. It is the measure of the current produced by an electromotive force of one volt through a resistance of one ohm.

Cathode. The terminal of an electric circuit from which the current flows towards the anode or positive pole, through a solution or electrolyte.

Compound Wound. A method of winding a generator or motor in which a shunt winding is used for the field magnets, and in which also a second winding of the magnet is placed in series with other currents.

Converter. This changes a continuous current to an alternating current.

Dyne. The force which can impart an acceleration of one centimetre per second to a mass of one gramme.

Erg. The work expended in moving a body through a distance of one centimetre against a force of one dyne.

Farad. This being too large for ordinary purposes, the microfarad is used in its place.

Frequency or Periodicity. The number of cycles per second of an alternating current.

Hysteresis. Magnet hysteresis is the property of those magnetic bodies in which the magnetic condition lags behind the magnetic force.

Induction Motor. An electric current flowing through a conductor in a magnetic field is capable of producing motion in the conductor or the magnet.

Monocycle System. This consists of a single-phase circuit for lighting. It can also be used for motors with the addition of an auxiliary or "teaser" to enable the motor to be started easily.

Multiphase or Polyphase Currents. A term applied to a group of currents of alternating type which constantly differ from each other by a constant proportion of periods of alternation.

Ohm. The resistance offered by a column of mercury one square millimetre in cross-section and 106·3 in length, at the temperature of melting ice.

Period. The time taken for a complete cycle of change in an alternating current.

Potential. The unit of difference existing between the end of a conductor when to cause the current to flow requires the expenditure of one erg per second.

Shunt Motor. A dynamo whose field is wound in shunt with the external circuit. Two leads are taken from the brushes; one goes round the field magnets to excite them, the other is the external circuit.

Synchronous Motor. An alternating-current motor that runs in exact accord with the varying phase or alternation of the current when the conductor or magnet is moved, producing an electric current.

Transformer. A transformer converts alternating currents from one voltage to another.

Electricity is much used in and about mines for many purposes, such as the motive power for various machines, hoists, locomotives, fans, mechanical shovels, slushers, pumping, also for signalling, illumination, exploders, electric furnaces, and the refining of metals, as in the case of copper.

The circuit or path a current takes may be in series or parallel. In a series circuit, the current passes through each of the units. The pressure furnished by the dynamo must be the sum of the pressures required to force the current through the different parts. Its objection for general use is that should a break occur at any point, it cuts off the current from flowing through the other parts. In such a system the dynamo is provided with an automatic regulator for the voltage. With the parallel circuit, the units are connected side by side in parallel across the main wires from the dynamo. The breaking of the circuit from any one of the units will not prevent the current from flowing through the other parts. This system is used for incandescent lamps; mine locomotives are also operated in parallel, the cars can be moved independently, and the current turned on and off at will. In the parallel system the pressure generated by the dynamo is kept constant. In mine haulage plants the pressure generally used is 250 volts, as being less dangerous than 500 volts, but on the surface 500 volts may be used.

The current may be direct or alternating. The direct or continuous current always passes in the same direction. Alternating currents are constantly changing in direction and value. Alternating currents may be single-, two-, or poly-phase; when one, two, or more conductors are used, the current passing by one and returning by the others; the difference in phase being a period or fraction of a period so that at a certain instant, say in the case of a three-phase, one line will be carrying a positive current equal in magnitude to the sum of the two negative currents in the other lines. When a direct-current motor is not otherwise in use, it can store energy in secondary batteries, which cannot be done with alternating currents.

To avoid line loss, it is important to have the copper wire of the proper size, for if it is too small it will overheat. The amount of current a given wire can carry without overheating largely depends on the location of the wire. If in the open air, it can carry a greater current than in boxed-up positions, when the heat is confined. The only way to keep down the current for a given amount of power is to increase the pressure. For a given amount of power transmitted, the weight of copper required decreases as the square of the voltage; so in order to transmit power for a long distance, a high pressure must be used. High tension lines on the surface are insulated on special porcelain insulators. In some cases bare wires are used for mine haulage, because ordinary insulation soon becomes defective underground.

A dynamo converts mechanical energy into electric energy. An electric motor converts electric energy into mechanical energy. Most motors used for mining purposes are run on constant potential system. Their efficiency value increases with the size of the machine; from 70 per cent for motors of ¾ to 1½ h.p. up to 90 per cent for motors of 15 h.p. and upwards.

KILOWATTS AND HORSE-POWER

0·746 Kilowatt equals One Horse-power

Kilowatts to Horse-power				Horse-power to Kilowatts			
kW	H.p.	kW	H.p.	H.p.	kW	H.p.	kW
1	1·341	55	73·733	1	0·746	55	41·03
2	2·681	60	80·436	2	1·492	60	44·76
3	4·022	65	87·139	3	2·238	65	48·49
4	5·363	70	93·842	4	2·984	70	52·22
5	6·703	75	100·545	5	3·730	75	55·95
6	8·044	80	107·248	6	4·476	80	59·68
7	9·384	85	113·951	7	5·222	85	63·41
8	10·725	90	120·654	8	5·968	90	67·14
9	12·065	95	127·357	9	6·714	95	70·87
10	13·406	100	134·048	10	7·460	100	74·60
11	14·747	110	147·47	11	8·206	110	82·06
12	16·087	120	160·87	12	8·952	120	89·52
13	17·428	130	174·28	13	9·698	130	96·98
14	18·768	140	187·68	14	10·444	140	104·44
15	20·109	150	201·09	15	11·190	150	111·90
16	21·450	160	214·50	16	11·936	160	119·36
17	22·700	170	227·90	17	12·682	170	126·82
18	24·131	180	241·31	18	13·428	180	134·28
19	25·471	190	254·71	19	14·174	190	141·74
20	26·812	200	268·12	20	14·920	200	149·20
22	29·493	220	294·93	22	16·412	220	164·12
24	32·174	240	321·74	24	17·904	240	179·04
26	34·856	260	348·56	26	19·396	260	193·96
28	37·537	280	375·37	28	20·888	280	208·88
30	40·218	300	402·18	30	22·380	300	223·80
32	42·899	325	435·69	32	23·872	325	242·45
34	45·580	350	469·21	34	25·364	350	261·1
36	48·261	400	536·24	36	26·856	400	298·4
38	50·943	450	603·27	38	28·348	450	335·7
40	53·624	500	670·30	40	29·840	500	373·0
42	56·305	600	804·36	42	31·332	600	447·6
44	58·986	700	938·42	44	32·824	700	522·2
46	61·667	800	1072·48	46	34·316	800	596·8
48	64·349	900	1206·54	48	35·808	900	671·4
50	67·030	1000	1340·60	50	37·300	1000	746·0

VOLTS LOST WITH VARIOUS COPPER WIRE COMBINATIONS

Sizes of Wires	Area, Circular Mils	Weight of Combination. Pounds per 1000 feet	Volts Lost per Ampere per 1000 feet
One No. 0000 and One No. 000	379,405	1149	0·027
,, ,, ,, 00	344,679	1044	0·030
,, ,, ,, 0	317,192	960	0·033
,, ,, ,, 1	295,295	894	0·035
One No. 0000 and Two No. 000	547,210	1657	0·019
,, ,, ,, 00	477,758	1447	0·022
,, ,, ,, 0	422,784	1280	0·025
,, ,, ,, 1	378,990	1148	0·027
One No. 000 and One No. 00	300,884	911	0·035
,, ,, ,, 0	273,397	828	0·038
,, ,, ,, 1	251,500	762	0·041
One No. 000 and Two No. 00	433,963	1314	0·024
,, ,, ,, 0	378,989	1147	0·027
,, ,, ,, 1	335,195	1015	0·031
One No. 00 and One No. 0	238,671	723	0·043
,, ,, ,, 1	216,774	656	0·048
One No. 00 and Two No. 0	344,263	1042	0·030
,, ,, ,, 1	300,469	910	0·035
One No. 0 and One No. 1	189,287	573	0·055
One No. 0 and Two No. 1	272,982	827	0·038

VOLTS DROP IN BONDED TRACK

Losses per Ampere per 1000 *Feet of Single Track*

Rail Bonds of No. 1 Copper Wire

Weight of Rail, Pounds per Yard	Length of Bond, Feet	Volts Drop per Ampere per 1000 feet of Single Track			
		30 ft Rail Lengths		20 ft Rail Lengths	
		Single-bonded	Double-bonded	Single-bonded	Double-bonded
8	2½	0·0344	0·0318	0·0354	0·0317
12	2½	0·0247	0·0221	0·0261	0·0224
16	2½	0·0198	0·0172	0·0214	0·0176
20	2½	0·0168	0·0142	0·0188	0·0149
25	2½	0·0145	0·0119	0·0166	0·0126
30	2½	0·0139	0·0108	0·0165	0·0120
35	3	0·0128	0·0097	0·0155	0·0109
40	3	0·0120	0·0089	0·0148	0·0101
45	3	0·0114	0·0083	0·0142	0·0096
50	3	0·0109	0·0078	0·0137	0·0090
55	3	0·0104	0·0073	0·0133	0·0086
60	3	0·0101	0·0070	0·0129	0·0083
65	3	0·0098	0·0067	0·0126	0·0080
70	3	0·0095	0·0064	0·0124	0·0077

Cables

There are three ways of lowering a cable down a shaft—

1. Lower the reel with the cable to the desired depth; then pull up the cable by means of the hoisting rope. This can only be done when the shaft or compartment is large enough for the reel to pass down.

2. Lash the cable to the hoisting rope at intervals, and lower with the main hoist. This cannot be done if the hoisting rope is flat.

3. Fix the reel on a substantial frame and provide two brake bands, one on each side of the reel. When the level along which it is to be conveyed is reached, clamp it, and draw the bight to the station.

A cable may be suspended in a vertical shaft by giving it three turns round a large hardwood roller soaked in paraffin and covering with soft rubber, the roller being suspended by heavy bolts from a beam at the head of the shaft. The two parts of the cable are then clamped together. The portion passing down the shaft is firmly attached to the side of the shaft on insulators at frequent intervals, while the upper portion is connected with heavy bar couplings to the outside feeders from which it can be readily disconnected. The trolley wires, which are bare, are arranged overhead either between the rails or over one rail according to the position of the trolley-pole. The supports are placed 25 to 40 ft apart. If the roadway is timbered, the insulators may be fastened to the cap pieces; if the roof is good they may be fixed in the rock, otherwise special supports must be erected. A good way to prevent men from coming in contact with the trolley wire is to surround it with a wooden box with the lower part on one side left open for the trolley pole to pass through. Mine waters are generally acid and when they come in contact with bare trolley wires, copper salts are formed round the wire, these gradually spreading over the surface of the insulator to neighbouring surfaces, creating a "ground." Rails are generally used for the return circuit. As the conductivity of iron is low, and fish-plates are liable to rust, the rails are bonded with heavy copper wire. Cross-connections between opposite rails should be made not less than every 150 ft.

Electric Locomotives

Electric locomotives for mines are generally designed to run 5 to 10 miles an hour. They range in size from 4000 lb in weight with a draw-bar pull of 500 lb, to 30,000 lb in weight with a 5500 lb draw-bar pull. The minimum gauge of rails is 18 in., but this is not to be recommended. The wheels may be inside or outside the cast-iron frame. Locomotives for underground work are generally made with two pairs of wheels so as to allow them to pass round small radius curves easily.

Electric locomotive haulage may be done by the ordinary trolley, or by storage batteries, and can be made to suit low or narrow entries. They can be used on temporary and crooked tracks where rope haulage would be impracticable. To haul skips out of bords, one may use either the storage battery locomotive; a cable reel type, in which case a cable that can be coiled up and let out from a drum attached to the main cable is required; or the crab type of locomotive, which remains at the mouth of the bord and hauls the skip out by a rope. Direct-current motors are found most suitable for underground haulage where the speed is comparatively slow, the run comparatively short, and there is much switching. For the same horse-power an alternating-current equipment is 25 to 35 per cent heavier, and consequently a greater dead weight has to be moved.

Drawbar Pull on Various Grades for Different-sized Locomotives

Horse-power	Weight	Level	Grades					
			1%	2%	3%	4%	5%	6%
10	4,000	500	460	420	380	340	300	260
20	8,000	1000	920	840	760	680	600	520
30	12,000	1500	1380	1260	1140	1020	900	780
50	20,000	2500	2300	2100	1900	1700	1500	1300
70	28,000	3500	3220	2940	2660	2380	2100	1820
100	40,000	5000	4600	4200	3800	3400	3000	2600

Electric hoists are rapidly taking the place of steam hoists, especially for deep shafts. They are not so large as steam winding engines. Their cost is less, so the interest on capital is less, too. The cost of energy per ton hoisted is less. There is a smaller consumption of oil, waste, and grease, and a lower cost for repairs and maintenance. There is an absence of stand-by losses. They are safe, reliable, and simply controlled. Hoisting by steam is very uneconomic; within a short period, say a minute, it has to start a heavy load slowly, accelerate, slow down, and stop.

The electric power may be developed at the mine, or purchased from some large central station. The high-speed condensing turbine has now become so economical and reliable, and occupies such a small space, that production of electric power by the turbo-generator and flywheel has become common practice.

There are two main types of electric hoists—

1. The induction motor system with Ward-Leonard control. The current from the generator is taken directly to the motor of the winder which is generally geared to the winding

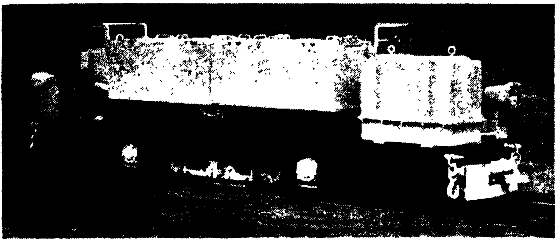


FIG. 5A. 10-TON BATTERY LOCOMOTIVE FOR UNDERGROUND TRANSPORTATION OF MEN AND MATERIALS
(*Courtesy of Metro-Vickers Electrical Co. Ltd.*)

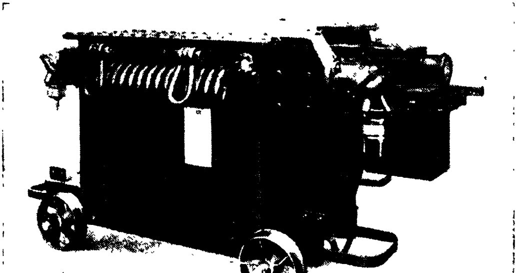


FIG. 5B. UNDERGROUND PORTABLE TRANSFORMER SUB-STATION
(*Courtesy of Metro-Vickers Electrical Co. Ltd.*)

drum shaft. The starting. stopping and speed regulation are effected through switches and resistances inserted in the electric current of the motor. The driving motor may be induction or synchronous which runs at a constant speed.

2. The Ilgner system. The current is not taken directly to the motor of the winder, but to a motor generator or flywheel set, and it is this which finally supplies the current to the main winder. The flywheel acts as a load equalizer.

Either of these systems can be used with direct or alternating current.

The Westinghouse converter equalizer system is intermediate between the other two. The power mains are connected directly to the winding motor, also through transformers to a rotary converter; this latter on the direct current side is connected to a direct current flywheel machine which acts as an equalizer. This machine acts as a generator when the load is above the average, and as a motor when the load falls below the average. In order to cause this change of action automatically, special transformers are inserted in the main transmitting lines between the power house and the connections to the transformers of the rotary converter, work an automatic regulator which varies the resistance of the field circuit of the direct-current machine.

Electricity is used as the motive power for driving either plunger or centrifugal pumps, and is useful for sinking pumps.

Batteries are used for bell signalling.

The electric exploder is a magneto machine worked either by turning a crank or by moving a rack-bar up or down. When a number of holes are to be exploded together, they are connected with wires in series if low tension fuses are used, or in parallel if high tension fuses are employed. Low tension fuses require 0·3 amp for each fuse but only a few volts, about 1·8. High tension fuses require 50 to 100 volts.

Arc lamps are used at stations, while incandescent lights are used along entries and drifts, so there is no necessity for men to wait till they get their underground sight after reaching their level before proceeding to work.

Safety Precautions

A competent person should always be on duty where electrical apparatus or machinery is in use. If the amount of electricity delivered underground exceeds 200 h.p., a competent person must be on duty above ground and another below ground.

Every person appointed to work an electrical apparatus must have been instructed in his duty and be competent for the work he has to do.

No unauthorized person should enter a motor-room, or interfere in any way with an electric line, machine, or apparatus.

Trailing cables used in connection with portable machines must be examined at least once each shift by the person in charge of the machine, and any defect in them promptly repaired.

The casing of inspection doors of all portable motors and the casings of their switches and other appliances should at least once a week be opened by a competent person, and the parts so disclosed shall be cleaned and examined before the coverings are replaced. In special cases requiring a motor to run continuously longer than one week, the motor should be examined at the end of the run.

If owing to any defect, any electric sparking or arc be produced outside a portable motor, or by the cable, or rails, the machine should be stopped and not worked again until the defect is repaired.

In underground roads, trolley wires must be so placed that they are at least 7 ft above the level of the road, or else sufficiently guarded, or the pressure must be cut off from the wires during such hours as the roads are used for travelling on foot in places where trolley wires are fixed. At other times no person other than a person duly authorized should be permitted to travel on foot along the road.

The person in charge of an electric machine of any sort underground must not leave it while it is working, and before leaving the working place he should see that the voltage is cut off and the trailing cable disconnected from the junction box.

A portable machine should not be repaired till the voltage is cut off from the machine and trailing cable disconnected from the junction box.

Any person finding inflammable gas should at once erect a danger fence and report the finding to the deputy of the district.

A storage-battery locomotive of an approved flame-proof type may be used in any part of a coal-mine where the percentage of inflammable gas in the general body of the air in such part is less than ¼ per cent. A storage-battery locomotive not being of an approved flame-proof type should not be used in any gassy place. An electric locomotive of the trolley system should not be used in any gassy place or in any dry and dusty place. It should only be operated on main roads ventilated by intake air.

Notices should be prominently exhibited at every entrance containing a trolley wire, stating that a trolley wire exists therein and that unauthorized persons are prohibited from travelling on roads equipped with a trolley wire except during periods which must be clearly set out in the notices.

No repair work of any kind should be carried out in any part of the roadway containing a trolley wire, unless such section of the trolley wire is first made dead, or unless the manager has given special permission in writing and there is a responsible mine official present all the time.

Haulage. Workmen should not be required to walk more than a mile underground. Trains running for the convenience of workmen should be under the entire control of a person appointed to accompany and have charge of the train. Workmen should not occupy any other part of the train but the seats provided.

All transport systems should be equipped with efficient brakes, and with a proper system of signalling.

Unless there is sufficient room for men to stand between a train and the sides of a plane in places where animal or mechanical haulage is used, refuge holes not less than 6 ft high, 3 ft wide, and 4 ft deep inside timbers should be provided if the speed exceeds more than four miles an hour. If a load is drawn by an animal, the refuge holes should not be more than 50 yd apart, or if mechanical power is employed the refuge holes should not be more than 20 yd apart. Refuge holes must be kept free from rubbish.

Where there are trolley wires overhead, they should be so protected that men walking under them cannot come in contact with them through the medium of any tools they may be carrying.

If sprags are used to stop skips by being pushed between spokes, they must have hand guards.

Couplings between skips should be double.

When skips have to pass up and down an incline, a hold-back or bank headstop should be provided, arranged on the brow of the incline to prevent skips from starting down hill before everything is ready. To prevent a skip or set of skips from running away when being hauled up an incline should the rope or a coupling break, a forked dragbar should be attached to the last skip, in such a manner that it is held off the ground till the disconnection takes place, when it digs into the ground and stops or derails the skip. Should a skip break away while being lowered down an incline, arrangements must be made to derail it or switch it on to a siding.

Electromotive Forces of various Cells

Batteries	+ Plate	Porous Cell		− Plate	Volts
Daniell	Zinc amalg.	Sulphuric acid 7½ to 1	Saturated solution of copper sulphate	Copper	1·079
,,	,,	,, 22 to 1	,, ,,	,,	0·978
,,	,,	,, ,,	Nitrate of copper saturated	,,	1·000
,,	,,	,, ,,	Sulphate of copper	,,	0·909
,, P.O.	,,	,, ,,	,, ,,	—	—
Standard	—	Sulphate of zinc saturated solution	Saturated solution	—	1·079
Grove	,,	Sulphuric acid 7½ to 1	Nitric acid (fuming)	Platinum	1·956
,,	,,	Salt water	Nitric acid, sp. gr. 1·33	,,	1·904
,,	,,	Sulphuric acid 22 to 1	,, ,,	,,	1·810
,,	,,	Sulphate of zinc	,, ,,	,,	1·672
Bunsen	,,	Dilute sulphuric acid	Nitric acid	Carbon	1·734
Smee	Zinc	Sulphuric 1, Water 7		Platinized silver	0·47
Walker	,,	,,	,,	,, carbon	0·65
Callan	Zinc amalg.	Dilute sulphuric acid	Nitric acid	Cast iron	1·700
Poggendorf	,,	,, ,,	Bichromate of potash	Carbon	1·796 2·028
Marié Davy	,,	Sulphuric acid 22 to 1	Paste of sulphate of mercury	,,	1·524
,,	,,	Dilute sulphuric acid	,, ,,	,,	1·33
Leclanché	,,	Solution of sal ammoniac	Binoxide of Manganese	,,	1·48
De la Rue	Zinc	Chloride of ammonium		Silver and Ag Cl	1·030
Skrivanov (pocket form)	,,	Solution 75 caustic potash to 100 water		,,	1·4 to 1·5
Becquerel	Zinc amalg.	Sulphate of zinc	Sulphate of lead	Lead	0·55

ELECTROMOTIVE FORCES OF VARIOUS CELLS (*continued*)

Batteries	+ Plate	Porous Cell		− Plate	Volts
Niaudet . .	Zinc amalg.	Common salt	Chloride of lime	Carbon	1·65
Duchemin . .	"	"	Perchloride of iron	Lead	1·541
" . .	Platinum	Dilute sulphuric acid	Dilute sulphuric acid	Platinum	1·79
Latimer Clark (standard cell)	Zinc amalg.	Sulphate of zinc	Paste of sulphate of mercury	Mercury	1·457
Howell's Manganese; internal res. = 1 ohm (Hockin)	"	Ammonic sulphate 25 g crystallized salt to 1 litre water	Sulphuric acid, 1 acid to 5 parts water	Carbon + MnO_2 + $MnSO_4$	2·04
Higgin's cascade, internal res. = 0·170 ohm	Zinc in mercury	Chromic acid	Sulphuric acid	Carbon	1·9
Thame's . .	Zinc	Dilute sulphuric acid	Nitric acid + CrO_2Cl_2	Carbon	2
Bennet's, internal res. = 5 ohm	"	K H O with distilled water	Damped with K H O and water	Iron can with iron borings	1·3
Lalande-Chaperon	Zinc amalg.	Caustic soda solution	Oxide of copper or copper scale	Iron	1
Faure's secondary battery	Lead plate coated with minium	Dilute sulphuric acid	Dilute sulphuric acid	Lead plate coated with minium	2·0–2·2
Sellon-Volckmar	Lead plate primed with minium	Sol. sulphuric acid, sp. gr. 1·100°	Sol. sulphuric acid, sp. gr. 1·100°	Lead plate primed with minium	2·15
Planté . .	Lead	Dilute sulphuric acid	Dilute sulphuric acid	Lead (spongy)	2·0–2·2

Note. These e.m.f.'s are 1·1 per cent too high, and should be multiplied by 0·9889, the ratio of the B.A. unit to the legal ohm.

HYDRAULICS

Greatest Velocities of Current close to the Bed of a Stream consistent with the Stability of the Material over which the Water Flows

	Ft/sec		Ft/sec
River mud, liquid earth	0·25	Broken stones . .	4·00
Common clay, loam .	0·50	Soft Schistose rock .	4·50
River sand . .	1·00	Rocks with distinct layers . . .	6·00
Gravel, size of beans .	1·50	Hard rocks . .	10·00
,, 1 inch . .	2·00		
Large shingle . .	3·00		

The mean velocities of current will be about 1⅓ times the velocities given above.

Table of the Theoretical Discharge of Water by Round Apertures of various Diameters, and under Different Heads of Water Pressure (Box)

Diam. in Inches	Head of Water in Inches								
	1	2	3	4	5	6	7	8	9
	Discharge in Gal/min								
1	4·7	6·6	8·1	9·4	10·5	11·5	12·4	13·3	14·1
2	18·8	26·4	32·4	37·6	42·0	46·0	49·6	53·2	56·4
3	42·2	59·4	72·9	84·6	94·5	103	112	120	127
4	75·2	106	130	150	168	184	198	213	225
5	117	165	203	235	262	287	310	332	352
6	169	237	291	338	378	414	446	479	507
7	230	310	397	460	514	563	607	652	691
8	301	422	518	601	672	736	793	851	902
9	381	534	656	761	850	931	1006	1077	1142
10	470	660	810	940	1050	1150	1240	1330	1411
12	676	952	1168	1353	1512	1656	1785	1915	2030
14	920	1241	1588	1842	2058	2254	2430	2606	2764
16	1203	1690	2074	2406	2688	2944	3174	3405	3610
18	1523	2138	2624	3045	3402	3726	4018	4309	4568
20	1880	2640	3240	3760	4200	4600	4960	5320	5640
22	2275	3194	3920	4550	5082	5566	6002	6437	6824
24	2704	3808	4672	5414	6048	6624	7140	7660	8120
30	4230	5940	7290	8460	9450	10350	11160	11970	12690
Vel. in ft/sec	2·32	3·275	4·01	4·63	5·18	5·67	6·13	6·55	6·95

Table of the Theoretical Discharge of Water (*continued*)

Diam. in Inches	Head of Water in Inches							
	10	12	14	16	18	20	22	24
	Discharge in Gal/min							
1	14·8	16·2	17·6	18·8	19·9	21	22	23
2	59·2	64·8	70·4	75·2	79·6	84	88	92
3	133	146	158	169	179	189	198	207
4	237	259	281	301	318	336	352	368
5	370	405	440	470	497	525	550	575
6	533	583	663	677	716	756	792	828
7	725	794	862	921	975	1029	1078	1127
8	947	1037	1126	1203	1273	1344	1408	1472
9	1199	1312	1425	1523	1612	1701	1782	1863
10	1480	1620	1760	1880	1990	2100	2200	2300
12	2134	2333	2534	2707	2865	3024	3170	3312
14	2901	3175	3450	3684	3900	4116	4312	4508
16	3789	4147	4506	4813	5094	5376	5632	5888
18	4795	5249	5702	6091	6447	6804	7128	7452
20	5920	6480	7040	7520	7960	8400	8800	9200
22	7163	7841	8518	9099	9632	10164	10648	11132
24	8536	9332	10136	10829	11460	12096	12680	13248
30	13320	14580	15840	16920	17910	18900	19800	20700
Vel. in ft/sec.	7·32	8·03	8·67	9·27	9·83	10·36	10·87	11·35

Example. Find the theoretical discharge, through an aperture 8 in. in diameter, under a head of 18 in. *Ans.* 1273 gal/min.

Discharge through an Orifice in a Thin Plate. The stream contracts at a distance from the plate of half the diameter of the orifice, and its diameter is 0·784, that of the orifice being 1.

$$G = \sqrt{(H \times d^2 \times 10)}$$

$$H = \left(\frac{G}{d^2 \times 10}\right)^2$$

$$d = \left(\frac{G}{\sqrt{(H \times 10)}}\right)^{\frac{1}{2}}$$

Discharge by Short Tubes when not less than twice the diameter of the orifice gives a greater discharge than thin plates.

$$G = \sqrt{(H \times d^2 \times 13)}$$

$$H = \left(\frac{G}{d^2 \times 13}\right)^2$$

$$d = \left(\frac{G}{\sqrt{(H \times 13)}}\right)^{\frac{1}{2}}$$

Friction of Long Pipes. To calculate the loss of head by friction—

Let d = diameter of pipe in inches; L = length of pipe in yards; H = head of water in feet; G = gallons per minute.

$$G = \left(\frac{(3d)^5 \times H}{L}\right)^{\frac{1}{2}}$$

$$H = \frac{G^2 \times L}{(3d)^5}$$

$$d = \left(\frac{G^2 \times L}{H}\right)^{\frac{1}{5}} \div 3$$

$$L = \frac{(3d)^5 \times H}{G^2}$$

Loss of Head by Bends. Let H = head due to change of direction in inches; r = radius of the bore of the pipe in inches; R = radius of the centre line of the bend in inches; ϕ = angle of bend in degrees; V = velocity of discharge in feet per second.

$$H = \left\{0{\cdot}131 + 1{\cdot}847 \times \left(\frac{r}{R}\right)^{\frac{7}{2}}\right\} \times \frac{V^2 \times \phi}{960}$$

$$V^2 = \frac{960 \times H}{\phi \times \left\{0{\cdot}131 + 1{\cdot}847 \times \left(\frac{r}{R}\right)^{\frac{7}{2}}\right\}}$$

Table of the Actual Discharge by Short Tubes of various Diameters, with Square Edges and under Different Heads of Water Pressure, being Eight-tenths of the Theoretical Discharge (Box)

Diam. in Inches	Head of Water in Inches																
	1	2	3	4	5	6	7	8	9	10	12	14	16	18	20	22	24
	Discharge in Gal/min																
1	3·76	5·28	6·48	7·52	8·4	9·2	9·9	10·6	11·3	11·8	13·0	14·1	15·0	15·9	16·8	17·6	18·4
2	15·04	21·12	25·9	30·1	33·6	36·8	39·7	42·6	45·1	47·4	51·8	56·3	60·2	63·7	67·2	70·4	73·6
3	33·8	47·5	58·3	67·7	75·6	82·4	89·6	96·0	101·6	106·4	116·8	126	135	143	151	158	166
4	60·2	84·8	104	120	130	147	158	170	180	189	207	225	241	254	269	282	294
5	93·6	132	162	188	210	230	248	266	282	296	324	352	376	398	420	440	460
6	135	190	223	270	302	331	357	382	406	426	466	530	542	573	605	634	662
7	194	248	318	368	411	450	486	522	553	580	636	689	737	780	823	862	902
8	241	338	414	481	538	589	634	681	722	758	829	901	962	1018	1075	1126	1178
9	305	427	525	609	680	745	805	863	914	959	1049	1140	1218	1290	1361	1426	1490
10	376	528	648	752	840	920	992	1064	1129	1184	1296	1408	1504	1592	1680	1760	1840
12	541	762	934	1082	1210	1325	1428	1532	1624	1707	1866	2027	2166	2292	2419	2536	2650
14	736	993	1268	1474	1646	1803	1944	2085	2211	2321	2540	2760	2947	3120	3293	3450	3606
15	846	1188	1458	1692	1890	2070	2232	2394	2288	2664	2916	3168	3384	3582	3780	3960	4140
16	962	1352	1659	1925	2150	2355	2539	2724	2888	3031	3318	3605	3850	4075	4301	4406	4710
18	1218	1710	2099	2436	2722	2981	3214	3447	3662	3836	4199	4562	4873	5158	5443	5702	5962
20	1504	2112	2592	3008	3360	3680	3968	4256	4512	4736	5184	5632	6016	6368	6720	7040	7360
22	1820	2552	3136	3640	4065	4452	4801	5149	5459	5730	6272	6814	7279	7705	8131	8518	8905
24	2163	3046	3737	4331	4838	5299	5712	6128	6496	6828	7465	8108	8663	9168	9676	10144	10598
30	3384	4752	5832	6768	7560	8280	8928	9576	10152	10656	11664	12672	13536	14328	15120	15840	16560

TABLE OF THE DISCHARGE OF PIPES BY PRONY'S FORMULA

$\frac{H \times d}{L}$	Velocity in Ft/sec	Diameter of the Pipe in Inches						
		1	1½	2	2½	3	3½	4
		Gallons Discharged per Minute						
0·00002402	0·025	0·0511	0 1150	0·2045	0·3196	0·4602	0·6260	0·8180
0·00005437	0·05	0·1022	0·2301	0·4091	0·6392	0·9204	1·252	1·636
0·00009108	0·075	0·1534	0·3450	0·6136	0·9588	1·381	1·878	2·454
0·0001341	0·100	0·2045	0·4602	0·8182	1·278	1·841	2·504	3·273
0·0001836	0·125	0·2556	0·5750	1·023	1·598	2·301	3·130	4·090
0·0002394	0·15	0·3067	0·6900	1·227	1·917	2·761	3·756	4·908
0·0003016	0·175	0·3578	0·8053	1·432	2·237	3·221	4·382	5·728
0·0003702	0·2	0·4090	0·9204	1·636	2·557	3·682	5·008	6·546
0·0004452	0·225	0·4601	1·035	1·841	2·876	4·142	5·634	7·363
0·0005266	0·25	0·5112	1·150	2·045	3·196	4·602	6·260	8·180
0·0006140	0·275	0·5624	1·265	2·250	3·515	5·062	6·886	9·000
0·0007080	0·3	0·6135	1·381	2·454	3·835	5·522	7·512	9·819
0·0008087	0·325	0·6646	1·496	2·659	4 154	5·982	8·138	10·64
0·0009154	0·35	0·7157	1·611	2·864	4·474	6·443	8·764	11·46
0·0010286	0·375	0·7669	1·726	3 068	4·794	6·903	9·390	12·27
0·0011480	0·4	0·8180	1·841	3 273	5·113	7·363	10·02	13·09
0·001274	0·425	0·8691	1·955	3·477	5·433	7·823	10·64	13·91
0·001406	0·45	0·9202	2·071	3·682	5·757	8·284	11·27	14·73
0·001545	0·475	0·9713	2·186	3·886	6·077	8·744	11·89	15·55
0·001690	0·5	1·023	2·301	4·091	6·392	9·204	12·52	16·37
0·002	0·55	1·125	2·531	4·500	7·031	10·12	13·77	18·00
0·00233	0·6	1·227	2·761	4·909	7·670	11·04	15·02	19·64
0·002693	0·65	1·329	2·901	5·318	8·309	11·96	16·28	21·27
0·003079	0·7	1·431	3·221	5·727	8·948	12·88	17·53	22·91
0·003490	0·75	1·533	3·450	6·136	9·588	13·81	18·78	24·54
0·003926	0·8	1·636	3·682	6·544	10·23	14·73	20·03	26·18
0·004388	0·85	1·738	3·912	6·954	10·86	15·65	21·29	27·82
0·004876	0·9	1·841	4·142	7·363	11·51	16·57	22·53	29·46
0·005928	1·0	2·045	4·602	8·182	12·78	18·41	25·04	32·73
0·00648	1·05	2·147	4·832	8·591	13·42	10·33	26·29	34·37
0·00708	1·1	2·249	5·062	9·000	14·06	20·25	27·54	36·00
0·007691	1·15	2·351	5·292	9·409	14·70	21·15	28·80	37·64
0·008338	1·2	2454	5·522	9·818	15·34	22·09	30·05	39·28
0·009	1·25	2·556	5·753	10·23	15·98	23·01	31·30	40·91
0·009694	1·3	2·658	5·983	10·64	16·62	23·93	32·55	42·55
0·010407	1·35	2·761	6·213	11·04	17·26	24·85	33·80	44·18
0·0115	1·4	2·863	6·443	11·45	17·90	25·77	35·06	45·82
0·01192	1·45	2·965	6·673	11·86	18·53	26·69	36·31	47·46
0·0127	1·5	3·067	6·900	12·27	19·18	27·61	37·56	49·08

TABLE OF THE DISCHARGE OF PIPES BY PRONY'S FORMULA (*continued*)

$\frac{H \times d}{L}$	Velocity in Ft/sec	Diameter of the Pipe in Inches						
		5	6	7	8	9	10	12
		Gallons Discharged per Minute						
0·00002402	0·025	1·278	1·841	2·504	3·272	4·142	5·113	7·362
0·00005437	0·05	2·556	3·682	5·008	6·544	8·284	10·23	14·72
0·00009108	0·075	3·834	5·523	7·512	9·816	12·43	15·34	22·09
0·0001341	0·100	5·113	7·363	10·02	13·09	16·57	20·45	29·45
0·0001836	0·125	6·390	9·205	12·52	16·36	20·71	25·57	36·81
0·0002394	0·15	7·668	11·05	15·02	19·63	24·85	30·67	44·17
0·0003016	0·175	8·947	12·88	17·53	22·95	28·99	35·79	51·53
0·0003702	0·2	10·23	14·73	20·03	26·18	33·13	40·91	58·90
0·0004452	0·225	11·50	16·57	22·54	29·45	37·28	46·02	66·26
0·0005266	0·25	12·78	18·41	25·04	32·72	41·42	51·13	73·62
0·0006140	0·275	14·06	20·25	27·54	36·00	45·56	56·25	80·98
0·0007080	0·3	15·34	22·09	30·05	39·27	49·70	61·36	88·35
0·0008087	0·325	16·62	23·93	32·55	42·54	53·84	66·46	95·71
0·0009154	0·35	17·89	25·77	35·06	45·81	57·98	71·59	103·1
0·0010286	0·375	19·17	27·61	37·56	49·08	62·13	76·69	110·4
0·0011480	0·4	20·45	29·45	40·06	52·36	66·27	81·81	117·8
0·001274	0·425	21·73	31·29	42·57	55·63	70·41	86·94	125·2
0·001406	0·45	23·01	33·13	45·07	58·90	74·55	92·03	132·5
0·001545	0·475	24·29	34·97	47·58	62·17	78·70	97·14	139·8
0·001690	0·5	25·57	36·82	50·08	65·45	82·83	102·3	147·2
0·002	0·55	28·12	40·50	55·09	72·00	91·12	112·5	162·0
0·00233	0·6	30·68	44·18	60·10	78·54	99·40	122·7	176·7
0·002693	0·65	33·23	47·86	65·10	85·08	107·7	132·9	191·4
0·003079	0·7	35·79	51·54	70·11	91·63	116·0	143·2	206·1
0·003490	0·75	38·34	55·23	75·12	98·16	124·3	153·4	220·9
0·003926	0·8	40·90	58·90	80·13	104·7	132·5	163·6	235·6
0·004388	0·85	43·46	62·59	85·14	111·3	140·8	173·8	250·3
0·004876	0·9	46·02	66·27	90·14	117·8	149·1	184·2	265·1
0·005928	1·0	51·13	73·63	100·2	130·9	165·7	204·5	294·5
0·00648	1·05	53·69	77·31	105·2	137·4	174·0	214·7	309·2
0·00708	1·1	56·24	80·09	110·2	144·0	182·2	224·9	324·0
0·007691	1·15	58·80	84·67	115·2	150·5	190·5	235·2	338·7
0·008338	1·2	61·36	88·36	120·2	157·1	198·8	245·4	353·4
0·009	1·25	63·91	92·04	125·2	163·6	207·1	255·7	368·1
0·009694	1·3	66·47	95·72	130·2	170·2	215·4	265·9	382·8
0·010407	1·35	69·02	99·40	135·2	176·7	223·6	276·1	397·6
0·01115	1·4	71·58	103·1	140·2	183·3	231·9	286·4	412·3
0·01192	1·45	74·14	106·8	145·2	189·8	240·2	296·6	427·0
0·0127	1·5	76·68	110·05	150·2	196·3	248·5	306·8	441·7

TABLE OF THE DISCHARGE OF PIPES BY PRONY'S FORMULA (*continued*)

$\frac{H \times d}{L}$	Velocity in Ft/sec	Diameter of the Pipe in Inches						
		14	15	16	18	20	21	24
		Gallons Discharged per Minute						
0·00002402	0·025	10·02	11·50	13·09	16·56	20·45	22·53	29·45
0·00005437	0·05	20·03	23·00	26·18	33·12	40·91	45·07	58·90
0·00009108	0·075	30·05	34·50	39·27	49·68	61·36	67·61	88·35
0·0001341	0·100	40·06	46·02	52·36	66·23	81·81	90·14	117·8
0·0001836	0·125	50·08	57·50	65·45	82·80	102·3	112·7	147·3
0·0002394	0·15	60·10	69·00	78·54	99·36	122·7	135·2	176·7
0·0003016	0·175	70·11	80·54	91·63	115·9	143·2	157·7	206·1
0·0003702	0·2	80·13	92·04	104·7	132·5	163·6	180·3	235·6
0·0004452	0·225	90·14	103·5	117·8	149·0	184·1	202·8	265·1
0·0005266	0·25	100·2	115·0	130·9	165·6	204·5	225·3	294·5
0·0006140	0·275	110·2	126·5	144·0	182·1	225·0	247·9	323·9
0·0007080	0·3	120·2	138·1	157·1	198·7	245·4	270·4	353·4
0·0008087	0·325	130·2	149·6	170·2	215·2	265·9	293·0	382·8
0·0009154	0·35	140·2	161·1	183·3	231·8	286·4	315·5	412·3
0·0010286	0·375	150·2	172·6	196·4	248·4	306·8	338·0	441·7
0·0011480	0·4	160·2	184·1	209·4	264·9	327·2	360·6	471·2
0·001274	0·425	170·3	195·6	222·5	281·5	347·7	383·1	500·6
0·001406	0·45	180·3	207·1	235·6	298·0	368·2	405·6	530·1
0·001545	0·475	190·3	218·6	248·7	314·6	388·6	428·2	559·5
0·001690	0·5	200·3	230·1	261·8	331·1	409·1	450·7	589·0
0·002	0·55	220·3	253·0	288·0	364·3	450·0	495·8	647·9
0·00233	0·6	240·4	276·1	314·2	397·4	490·9	540·8	706·8
0·002693	0·65	260·4	299·1	340·3	430·5	531·8	585·9	765·7
0·003079	0·7	280·5	322·1	366·5	463·6	572·7	631·0	824·6
0·003490	0·75	300·5	345·0	392·7	496·8	618·6	676·0	883·5
0·003926	0·8	320·5	368·2	418·9	529·8	654·5	721·1	942·4
0·004388	0·85	340·5	391·2	445·1	563·0	695·4	766·2	1001
0·004876	0·9	360·6	414·2	471·2	596·1	736·8	811·3	1060
0·005928	1·0	400·6	460·0	523·6	662·3	818·1	901·4	1178
0·00648	1·05	420·6	483·0	549·8	695·4	859·0	946·5	1237
0·00708	1·1	440·6	506·0	576·0	728·5	900·0	991·6	1296
0·007691	1·15	460·7	529·0	602·1	761·6	940·9	1037	1355
0·008338	1·2	480·7	552·2	628·3	794·8	981·9	1082	1414
0·009	1·25	500·8	575·2	654·5	827·9	1023	1127	1472
0·009694	1·3	520·8	598·2	680·7	861·0	1064	1172	1531
0·010407	1·35	540·8	621·3	706·9	894·1	1104	1217	1590
0·01115	1·4	560·9	644·3	733·0	927·2	1145	1262	1649
0·01192	1·45	580·9	667·3	759·2	960·3	1186	1307	1708
0·0127	1·5	601·0	690·0	785·4	993·6	1227	1352	1767

TABLE SHOWING THE PROPER ANGLE FOR THE SIDES OF AQUEDUCTS OF DIFFERENT MATERIALS, AND THE RATE OF FLOW THAT SHOULD NOT BE EXCEEDED

Nature of Sides of Aqueduct	Angle of Sides	Maximum Rate of Flow
	Degrees	
Brickwork, masonry, or solid rock	90	5 to 10 ft/sec
Stone without mortar	60	4 ft/sec
Clay	45	
Coarse gravel	40	3 ft/sec
Fine gravel	35	2 ft/sec
Sand	30	1 ft/sec
Ordinary soil	20	3 in/sec.

Water flows quicker over hard than soft material. The flow should be at least 1 to 1½ ft/sec to prevent deposits or growth of plants.

TABLE SHOWING VELOCITIES OF WATER IN CHANNELS IN EARTH FOR VARIOUS SLOPES AND HYDRAULIC MEAN DEPTHS, THE HYDRAULIC MEAN DEPTH BEING THE SECTIONAL AREA OF WATERWAY DIVIDED BY THE WETTED BORDER OF THE CHANNEL

Hydraulic Mean Depth	Fall per Mile							
	6 in.	9 in.	12 in.	18 in.	24 in.	30 in.	36 in.	48 in.
0·50					36·7	41·3	45·5	52·8
0·75			35·0	43·6	50·8	57·2	62·8	72·9
1·00		37·7	44·1	54·8	63·7	71·6	78·6	91·0
1·25	36·2	45·1	52·6	65·1	75·6	84·9	93·2	108·0
1·50	41·8	52·0	60·6	74·8	86·8	97·3	106·8	123·7
1·75	47·3	58·6	68·1	84·0	97·4	109·1	119·7	
2·00	52·5	64·9	75·3	92·8	107·5	120·4		
2·50	62·4	76·8	88·9	109·2	126·3			
3·00	71·6	87·8	101·8	124·5				
3·50	80·4	98·3	113·5					
4·00	88·7	108·3	124·8					
5·00	104·4	126·9						
6·00	118·9							

TABLE OF THE VELOCITIES OF DISCHARGE IN OPEN CHANNELS, RIVERS, ETC., WITH DIFFERENT HEADS

Hydraulic Radius in Feet	Fall in Ft/m and In./yd															
	2	3	4	5	6	7	8	9	10	12	15	20	25	30	40	50
	0·01364	0·02045	0·02727	0·03409	0·04091	0·04773	0·05454	0·06136	0·06818	0·08182	0·1023	0·1364	0·1704	0·2045	0·2727	0·3409
	Mean Velocity throughout the whole Cross-sectional Area in Ft/min															
0·1	34·5	42·3	48·8	54·6	59·8	64·6	69·1	73·2	77	84	94	109	122	134	154	173
0·2	48·8	59·8	69·1	77·2	84·6	91·4	97·7	103	109	120	134	154	173	189	218	244
0·3	59·8	73·2	84·6	94·6	104	112	120	127	134	146	164	189	211	232	268	299
0·4	69·1	84·6	97·7	109	120	129	138	146	154	169	189	218	244	267	308	345
0·5	77·2	94·6	109	122	134	144	154	164	173	189	211	244	272	299	346	386
0·6	84·6	103·6	120	134	147	158	169	179	189	207	232	268	299	328	378	423
0·7	91·4	111·9	129	144	158	171	183	194	204	224	250	289	323	354	408	457
0·8	97·7	119·6	138	154	169	183	195	207	218	239	267	309	345	378	436	488
0·9	103·6	126·9	146	164	179	194	207	220	232	254	284	328	366	401	464	518
1·0	109·2	133·7	154	173	189	204	218	232	244	267	299	345	386	423	488	546
1·1	114·5	140·3	162	181	198	214	229	243	256	280	314	362	405	444	512	573
1·2	119·6	146·5	169	189	207	224	239	254	267	293	328	378	423	463	534	598
1·3	124·5	152·5	176	197	216	233	249	264	278	305	341	394	440	482	556	623
1·4	129·2	158·3	183	204	224	242	258	274	289	316	354	409	457	500	578	646
1·5	133·8	163·8	189	211	232	250	267	284	299	328	366	423	473	518	598	669
1·6	138·1	169·2	195	218	239	258	276	293	309	338	378	437	488	535	618	691
1·7	142·4	174·4	201	225	247	266	285	302	318	349	390	450	503	551	636	712
1·8	146·5	179·4	207	232	254	274	293	311	327	359	401	464	518	567	654	733
1·9	150·5	184·4	213	238	261	282	301	319	336	369	412	476	532	583	672	753

Hydraulic Radius in Feet	Fall in Ft/m and In./yd															
	2	3	4	5	6	7	8	9	10	12	15	20	25	30	40	50
	0·01364	0·02045	0·02727	0·03409	0·04091	0·04773	0·05454	0·06136	0·06818	0·08182	0·1023	0·1364	0·1704	0·2045	0·2727	0·3409
	Mean Velocity throughout the whole Cross-sectional Area in Ft/min															
2·0	154·4	189·1	218	244	267	289	309	328	345	378	423	488	546	598	690	772
2·2	162·0	198·4	229	256	281	303	324	344	362	397	444	512	573	628	724	810
2·4	169·2	207·2	239	267	293	316	338	359	378	414	463	535	598	655	756	846
2·6	176·1	215·6	249	278	305	329	352	374	394	431	482	557	623	682	788	880
2·8	182·7	223·8	258	289	317	342	365	388	409	448	500	578	646	708	818	914
3·0	189·2	231·7	267	299	328	354	378	401	423	463	518	598	669	733	846	946
3·2	195·4	239·3	276	309	338	365	391	414	437	478	535	618	691	756	874	977
3·4	201·4	246·6	285	318	349	377	403	427	450	493	551	637	712	780	900	1007
3·6	207·3	253·8	293	328	359	388	414	440	463	508	567	655	733	803	926	1036
3·8	212·8	260·7	301	337	369	398	425	452	476	522	583	673	753	824	952	1063
4·0	218·2	267·5	309	345	378	409	436	463	488	535	598	691	772	846	976	1091
4·4	229·0	280·5	324	362	397	429	458	486	512	561	627	724	810	887	1024	1145
4·8	239·2	293·2	338	378	414	448	478	508	535	586	655	757	846	926	1070	1196
5·2	249·0	305·2	352	394	431	466	498	528	557	610	682	788	881	964	1114	1245
5·6	258·4	316·6	365	409	448	483	517	548	578	633	708	817	914	1001	1156	1292
6·0	267·6	327·6	378	423	463	500	535	567	598	655	733	846	946	1036	1196	1338
7·0	288·8	353·9	408	457	500	540	577	611	646	708	791	914	1019	1119	1292	1444
8·0	308·8	378·3	436	488	535	578	618	655	691	757	846	977	1091	1196	1382	1544
9·0	327·6	401·3	463	518	567	613	655	695	733	803	896	1036	1158	1269	1466	1638
10·0	345·4	422·9	488	546	598	646	691	733	772	846	946	1092	1221	1337	1544	1727

Velocity of Efflux. V = velocity in feet per second; H = head of water in feet, then—$V = \surd(H \times 8)$.

Discharge in Gallons. G = gallons discharged per minute; H = head of water in feet; d = diameter of orifice in inches, then—$G = \surd(H \times d^2 \times 16{\cdot}3)$.

The above being the *theoretical* velocities discharged require modification according to the particular form of orifice.

Effect of Contour of Section. The hydraulic mean gradient corresponds to a straight line drawn between the points of entry and exit of a pipe. No loss of effect will arise from the pipe following the section of the ground, so long as the contour of the pipe does not anywhere along the line rise above the hydraulic mean gradient. Where it is necessary to conduct water over a hill above the hydraulic mean gradient, but below the top of the pipe, the first section of piping, having a low head, must be of a greater diameter than the subsequent section, which has a greater head.

General Laws for Pipes. When the diameter and length are constant, the discharge varies directly as the square root of the head. Conversely, the head is directly as the square of the discharge.

When the head and length are constant, the discharge is directly as the 2·5 power of the diameter. Conversely, the diameter will vary directly as the 2·5 root of the discharge.

When the discharge and length are constant, the head will be inversely as the 5th power of the diameter. Conversely, the diameter will be inversely as the 5th root of the head.

When the head and diameter are constant, the discharge will be inversely as the square root of the length. Conversely, the length varies inversely as the square of the discharge.

When discharge and diameter are constant, the head is directly and simply as the length.

Head for Very Low Velocities (*Prony's Rule*). Let d = diameter of the pipe in inches; H = head of water in inches; L = length of pipe in feet; G = gallons per minute.

$$G = \left(16{\cdot}353 \times \frac{H \times d}{L} + 0{\cdot}00665\right)^{\frac{1}{2}} - 0{\cdot}0816 \times d^2 \times 2{\cdot}04$$

$$H = \frac{\left(\frac{G}{2{\cdot}04 \times d^2} + 0{\cdot}0816\right)^2 - 0{\cdot}00665 \times \frac{L}{d}}{16{\cdot}353}$$

Head Due to Velocity in Open Channels. When a stream leaves a reservoir by a channel the bottom of which is level with that of the reservoir, and its entrance is well rounded, the velocity is 0·96 of that due to gravity; but if the sides of the inlet opening are square, the velocity will be 0·86 of that due to gravity. With an opening in a sluice gate of small thickness, the head of water being above the lower edge of the gate, the velocity is only 0·635 of that due to gravity. In the case of a weir, the contraction occurring on three sides only, the coefficient rises to 0·667.

TABLE OF THE VELOCITIES IN FEET PER SECOND DUE TO GIVEN HEADS (BOX)

Head in In.	A Coef. 1·0	B Coef. 0·96	C Coef. 0·86	D Coef. 0·635	Head in In.	A Coef. 1·0	B Coef. 0·96	C Coef. 0·86	D Coef. 0·635
1/64	0·29	0·2784	0·2494	0·18415	1	2·317	2·2224	1·9980	1·4713
1/32	0·41	0·3936	0·3524	0·2603	1¼	2·590	2·4864	2·2270	1·6446
1/16	0·58	0·5568	0·4988	0·3683	1½	2·837	2·7235	2·4398	1·8015
1/8	0·82	0·7872	0·7052	0·5207	1¾	3·065	2·9424	2·6360	1·9463
3/16	1·0	0·9600	0·8600	0·6350	2	3·276	3·145	2·8174	2·0803
1/4	1·158	1·1117	0·9959	0·7353	2¼	3·475	3·336	2·9885	2·2066
5/16	1·295	1·2432	1·1140	0·8223	2½	3·663	3·516	3·1502	2·3260
3/8	1·418	1·3613	1·2195	0·9004	2¾	3·842	3·688	3·3041	2·4397
7/16	1·532	1·4707	1·3175	0·9728	3	4·012	3·851	3·4503	2·5476
1/2	1·638	1·5725	1·4087	1·0401	3¼	4·176	4·009	3·5914	2·6517
9/16	1·737	1·6675	1·4938	1·1030	3½	4·334	4·161	3·7272	2·7521
5/8	1·831	1·7577	1·5747	1·1627	3¾	4·486	4·306	3·8580	2·8486
11/16	1·921	1·8442	1·652	1·2198	4	4·633	4·448	3·9844	2·9420
3/4	2·006	1·9258	1·725	1·2738	4½	4·914	4·717	4·2260	3·1204
13/16	2·088	2·0045	1·796	1·3259	5	5·180	4·973	4·455	3·2893
7/8	2·167	2·0803	1·863	1·376	5½	5·433	5·216	4·672	3·450
15/16	2·243	2·1533	1·929	1·424	6	5·675	5·448	4·881	3·6036

Discharge of Water Courses. Find the maximum central surface velocity by means of a float, take 84 per cent of this for the mean velocity throughout the section, multiply this by the sectional area for the discharge per cubic foot.

Head to Overcome Friction of Channel. Let L = length of channel in yards; A = cross-sectional area of the stream in square feet; P = the perimeter, or wetted border, in feet; F = the fall, or difference of level at the two ends of the channel, in inches; C = cubic feet discharged per minute.

$$F = \frac{\left(\frac{C}{A}\right)^2 \times L \times P}{874520 \times A}; \quad C = \left(\frac{874520 \times F \times A}{L \times P}\right)^{\frac{1}{2}} \times A$$

Submerged Openings. The velocity of discharge through a submerged opening is governed by the difference of the level of water at the two sides of it, or by the head, and is not affected by the depth below the surface at which it is placed.

Head for Low Velocities

Let V = mean velocity over the whole area in feet per second;

$$R = \text{the hydraulic radius in feet, or } \frac{\text{area in square feet}}{\text{border in feet}};$$

$$S = \text{the slope, or } \frac{\text{fall in inches}}{\text{length in inches}}$$

$$\frac{(V + 0{\cdot}1089)^2 - 0{\cdot}0118858}{8975} = RS.$$

To find the Velocity—

$$\frac{\text{Area of channel in square feet} \times \text{fall in inches}}{\text{Border in feet} \times \text{length of the channel in inches}}$$

find the nearest number thereto in column B of the following Table, and opposite to that number in column A is the required velocity.

To find the Fall in inches—

$$\frac{\text{The given discharge}}{\text{The given area} \times 60} = \text{mean velocity in feet per second}$$

$$\frac{\text{Nearest number to velocity in feet per second in column A} \times \text{border in feet} \times \text{length of channel in inches}}{\text{area in square feet}}$$

TABLE FOR THE DISCHARGE OF CANALS, RIVERS, ETC., BY EYTELWEIN'S RULE

Mean Velocity in Ft/sec	RS	Mean Velocity in Ft/sec	RS	Mean Velocity in Ft/sec	RS
0·025	0·0000006734	0·375	0·00002477	0·95	0·0001236
0·05	0·000001489	0·4	0·00002753	1·0	0·0001357
0·075	0·00000244	0·425	0·00003043	1·1	0·00016146
0·1	0·000003538	0·45	0·000033484	1·2	0·0001895
0·125	0·000004771	0·475	0·00003666	1·3	0·00021984
0·15	0·000006144	0·5	0·00003998	1·4	0·0002524
0·175	0·000007656	0·55	0·00004705	1·5	0·00028703
0·2	0·000009307	0·6	0·00005466	1·6	0·00032402
0·225	0·0000111	0·65	0·00006284	1·7	0·0003632
0·25	0·00001303	0·7	0·00007158	1·8	0·0004047
0·275	0·00001510	0·75	0·00008087	1·9	0·000448
0·3	0·00001730	0·8	0·00009072	2·0	0·0004943
0·325	0·00001966	0·85	0·00010112	2·5	0·000757
0·35	0·00002214	0·9	0·0001121	3·0	0 001075
A	B	A	B	A	B

GEOLOGY

Content of the Crust of the Earth

Percentage of Sundry Elements

Silicon	27·720
Aluminium	5·010
Iron	5·010
Calcium	3·630
Sodium	2·850
Potassium	2·600
Magnesium	2·090
Titanium	0·630
Manganese	0·100
Barium	0·050
Chromium	0·037
Carbon	0·032
Zirconium	0·026
Nickel	0·020
Vanadium	0·017
Cerium and Yttrium	0·015
Copper	0·010
Tungsten	0·005
Lithium	0·004
Zinc	0·004
Columbium and Tantalum	0·003
Hafnium	0·003
Lead	0·002
Cobalt	0·001
Boron	0·001
Beryllium	0·001
Molybdenum	0·0001
Arsenic	0·0001
Tin	0·0001
Cadmium	0·00001
Mercury	0·00001
Silver	0·000001
Selenium	0·000001
Gold	0·0000001

Main Geological Periods

Quaternary	Recent or Prehistoric Pleistocene	Age of Man
Cainozoic	Tertiary Pilocene	Age of mammals
	Miocene Oligocene Eocene	Evolution of primates
Mesozoic	Cretaceous	Culmination of reptiles
	Jurassic	Evolution of birds, age of reptiles
	Triassic	Evolution of mammals, increase of reptiles
Upper Paleozoic	Permian Carboniferous	Evolution of reptiles, age of amphibians, age of coal
	Devonian	Evolution of amphibians, age of fishes
	Silurian	Development of fishes
Lower Paleozoic	Ordovician	Evolution of vertebrates
	Cambrian	Reign of invertebrates
Proterozoic	Algonkian Archaean	Evolution of invertebrates

These have various subdivisions peculiar to different districts which merge one into the other.

Igneous Rocks

Igneous rocks have been classified in many different ways: according to their mineral composition; structure (granular, porphyritic, glassy); mode of occurrence (massive, dykes, surface flows, fragmental), etc. For the miner, perhaps the best classification would be according to the mineral composition, subdivided into structure, though the constituent minerals are subject to modifications, and at times the structures run into one another, so there are to a certain extent intermediate rocks. There are many names given to varieties owing to some slight difference from the type to which they belong which are often purely of local interest and may require the help of a microscope to determine them. The structure depends to a large extent on the rate at which the rock cooled. Glassy rocks such as obsidian, pitchstone, and tachylite cooled quickly, before the minerals had time to separate out. Lava flows have taken longer and are fine-grained, sometimes showing flow structure, and cavities. Masses and dyke rocks have retained their heat longer, so they have coarser grains, easily distinguished by the naked eye. When a rock with either a fine or coarse grain matrix has well-formed crystals that have separated out, it is said to be porphyritic.

The common rock-forming minerals are quartz, felspar (soda and potash felspars), mica, pyroxene, hornblende, nepheline, leucite, together with accessory minerals, e.g. tourmaline, olivine, apatite, epidote, magnetite, ilmenite, pyrites, pyrrhotite, etc.

Granite. The essential minerals are quartz, felspar, and generally mica, with hornblende, pyroxene, or apatite as accessory minerals. It is light-coloured grey, reddish, or greenish. When felspar crystals are well developed the rock is known as felspar porphyry, or if the crystals developed are quartz, it is called quartz porphyry.

Apilite, also known as elvan, is a dyke rock of finer grain than granite.

Pegmatite or *Giant granite* occurs in veins, the minerals forming it are in large masses.

Rhyolite is a fine form of porphyry which shows flow structure.

Syenite may be looked on as a quartzless granite in which the mica has been replaced by hornblende and sometimes by augite.

Trachyte is the flow form of syenite.

Monzonite comes half-way between granite and diorite: it is mainly felspar with hornblende, pyroxene, or mica, but seldom with quartz.

Diorite. The constituent minerals are felspar and certain dark minerals, e.g. hornblende, mica, and sometimes pyroxene, but seldom quartz. It may be granular or porphyritic. It is dark, generally greenish in colour and mottled. Andesite is the dyke or flow form of diorite. It has a fine porphyritic structure and is dark grey or greenish brown in colour. Accessory minerals are magnetite, ilmenite, pyrite, and epidote.

TABLE SHOWING THE UNDERLIE AND PERPENDICULAR IN FEET AND INCHES TO EVERY DEGREE OF THE QUADRANT, IN SIX FEET = ONE FATHOM

Deg.	Base		Perpendicular		Deg.	Deg.	Base		Perpendicular		Deg.
	Ft	In.	Ft	In.			Ft	In.	Ft	In.	
0	6	0	0	0	**90**	**23**	5	6¼	2	4	**67**
1	6	0	0	1¼	**89**	**24**	5	6	2	5¼	**66**
2	5	11¾	0	2½	**88**	**25**	5	5¼	2	6⅜	**65**
3	5	11⅔	0	3¾	**87**	**26**	5	4¾	2	7½	**64**
4	5	11⅝	0	5	**86**	**27**	5	4⅛	2	8⅝	**63**
5	5	11½	0	6⅛	**85**	**28**	5	3⅝	2	9⅞	**62**
6	5	11⅜	0	7½	**84**	**29**	5	3	2	11	**61**
7	5	11¼	0	8⅝	**83**	**30**	5	2⅜	3	0	**60**
8	5	11⅛	0	10	**82**	**31**	5	1¾	3	1	**59**
9	5	11	0	11	**81**	**32**	5	1	3	2⅛	**58**
10	5	10⅞	1	0¼	**80**	**33**	5	0½	3	3¼	**57**
11	5	10⅝	1	1¾	**79**	**34**	4	11¾	3	4⅛	**56**
12	5	10⅜	1	3	**78**	**35**	4	11	3	5¼	**55**
13	5	10⅛	1	4	**77**	**36**	4	10¼	3	6¼	**54**
14	5	9⅞	1	5⅜	**76**	**37**	4	9½	3	7¼	**53**
15	5	9⅝	1	6½	**75**	**38**	4	8⅞	3	8¼	**52**
16	5	9¼	1	8	**74**	**39**	4	8	3	9⅛	**51**
17	5	8⅞	1	9	**73**	**40**	4	7¼	3	10⅛	**50**
18	5	8½	1	10¼	**72**	**41**	4	6½	3	11⅛	**49**
19	5	8⅛	1	11⅜	**71**	**42**	4	5⅝	4	0	**48**
20	5	7¾	2	0½	**70**	**43**	4	4¾	4	1	**47**
21	5	7¼	2	1¾	**69**	**44**	4	4	4	2	**46**
22	5	6¾	2	3	**68**	**45**	4	3	4	3	**45**
	Ft	In.	Ft	In.			Ft	In.	Ft	In.	
Deg.	Perpendicular		Base		Deg.	Deg.	Perpendicular		Base		Deg.

This table is calculated for the quadrant taking its angles from the horizon.

When the angle exceeds 45°, read from foot of table and take base and perpendicular marked there.

EXAMPLE. Angle 25°, length 9 fm 3 ft. See opposite 25°, 5 ft 5¼ in. × 9 fm 3 ft = 51 ft 7⅞ in. for base and 2 ft 6⅜ in. × 9 fm 3 ft = 24·0$\frac{1}{16}$ fm perpendicular.

Diabase, sometimes called *dolerite*. The chief constituent minerals are felspar and pyroxene; olivine, hornblende, magnetite, and ilmenite are often present. Its colour is green of various shades to almost black. It may be granular or porphyritic. Basalt is the lava of diabase.

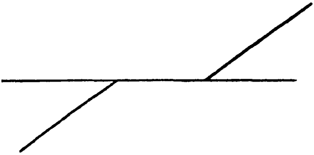


FIG. 6. NORMAL FAULT

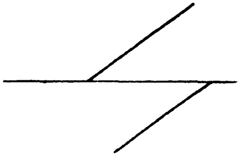


FIG. 7. REVERSE FAULT

Peridotite. The chief constituent minerals are olivine, pyroxene, and hornblende. Accessory minerals are magnetite, pyrrhotite, ilmenite; no felspar or quartz. Colour very dark green, almost black.

Greenstone is a general name for fine grain dark green rocks (diabase, diorite, andesite) which cannot be distinguished on the field.

Faults

Rocks are broken up by joints and more extensive fractures because they can no longer withstand the strains to which they are subjected. Such fractures do not take place without a certain amount of displacement, and such displacements are known as faults. There are two kinds of faults: normal, and reverse or overlap (Figs. 6 and 7). A lateral movement is called a heave; a vertical movement is termed a throw. We seldom if ever get a pure heave or throw, the two being combined: but one may predominate over the other. Though a heave may push a vein out of its course where it is crossed, the displacement is only local, for the vein away from the fault continues on its general strike (Fig. 8). When a fault takes place, both sides move, but in opposite directions. It is possible for a pure vertical down throw (if such a thing

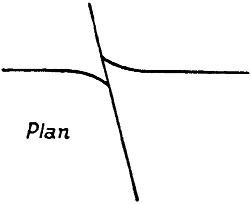


FIG. 8. FAULT: LOCAL DISPLACEMENT

exists) may have an apparent heave to the right or left depending on the underlie of the faulted vein (Fig. 9). When a vein is faulted, it is important to know in which direction to seek it after passing through the fault; and, on finding the lost portion, if the ore previously worked is not there, to know whether to sink or rise to discover it.

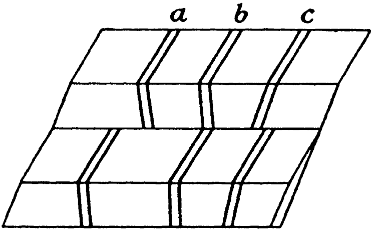


FIG. 9. APPARENT HEAVE DUE TO UNDERLIE
(*a*) To the left; (*b*) Not heaved; (*c*) To the right

Faults may be classified as strike faults, which run parallel with the vein or strata that they fault; or dip faults which cut across the vein or strata at various angles. A highly-inclined faulted vein is more easily recovered for working than a faulted horizontal bed; as in the former case one can continue to drive on the same level to find the lost portion;

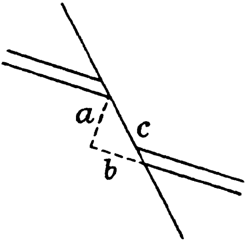


FIG. 10. NORMAL FAULT
(*a*) Vertical displacement
(*b*) Horizontal displacement
(*c*) Slip or throw

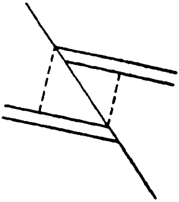


FIG. 11. REVERSE FAULT SHOWING OVERLAP

whereas, in the latter case, the lost portion can only be recovered by altering the level of the roadway, which may entail considerable expense. The vertical displacement of a fault (Fig. 10) is the perpendicular distance between the horizons of the two parts of the vein or bed. The offset is the horizontal distance between the two faulted ends of the lode. In a normal fault, this part has no lode in it, but in a reverse fault (Fig. 11) the lode is doubled. The slip or throw is measured along the dip of the fault. The displacement of a vein is not uniform throughout the length of the fault, being greater about the centre (Fig. 12). Care must be exercised when interpreting a fault seen on the surface, for in some cases the upper portion of a bed and the fault may have been denuded (Fig. 13) and the two faulted portions may be mistaken for two separate beds. If a fault cuts diagonally across a vein, the broken ends will be bevelled, which must not be mistaken for the dying out of the vein (Fig. 14). One must also be careful not to mistake unconformity for a fault

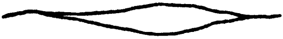


FIG. 12. A FAULT TAPERS OFF AT ITS END

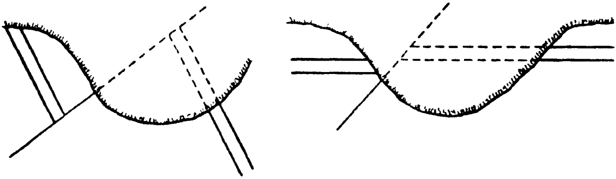


FIG. 13. A FAULTED BED MAY BE MISTAKEN FOR TWO SEPARATE BEDS ON ACCOUNT OF A PORTION OF THE FAULT AND BED HAVING BEEN DENUDED

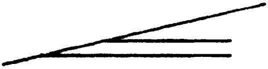


FIG. 14. END OF FAULTED VEIN BEVELLED

(Fig. 15) or a branch vein for part of a faulted vein (Fig. 16).

There are various rules for determining the direction in which to drive in order to find the lost portion of a vein or bed, whether heaved or thrown. These answer all right for normal faults, but not for reverse faults. Zimmermann's rule is a graphic one, and holds good for both normal and reverse faults. The only case in which it may appear wrong is when more than one fault is present which faults those first formed and there has not been sufficient work done to determine the order in which they occurred, so that one can work back (Figs. 17, (*a*) (*b*) (*c*)). Make a plan of the fault and known part of the vein, showing their respective strikes and dips. Draw a line where the fault and vein intersect at the upper and lower levels: produce this line past the upper level. Draw a line at the point of intersection at right angles to the fault. Whichever side of the produced line of intersection the perpendicular is found is the direction in which to drive after passing through the fault. In other words, look for the lost portion of the vein in the direction of the greater angle between the produced line of intersection and the fault, after having passed through the fault. If one has to drive on the right side to reach the lost lode, it is called a right-hand heave; if on the left, it is called a left-hand heave.

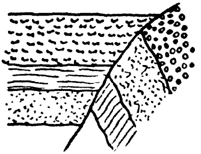


FIG. 15. UNCONFORMITY

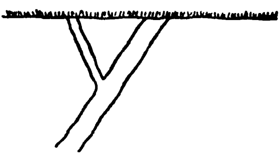
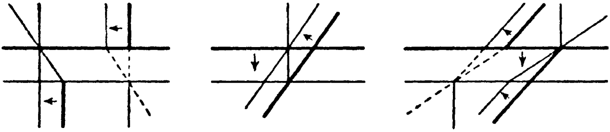


FIG. 16. BRANCH VEIN



(*a*) (*b*) (*c*)

FIG. 17. ZIMMERMANN'S RULE

(*a*) Right-hand heave; (*b*) Not heaved; (*c*) Left-hand heave

Fault rules give no indication as to how far one has to drive to pick up a faulted vein, but in some cases this may be determined if the wall rocks have some lithological peculiarities or certain strata have characteristic fossils (Fig. 18 (*a*)).

The following are rules for finding the direction to seek a bed that has been thrown up or down. Follow the trail of ore in the fault, if any, between the two parts of the faulted bed. This trail is more likely to occur if the fault was formed slowly and under pressure. Follow the bending of the vein at the fault (Fig. 18 (*b*)). Rocks which are plastic show evidence of bending before fracturing, the vein feathering out instead of being cut off suddenly. In normal cases the bending takes place in the direction of the throw of the fault. In a reverse fault this does not hold good, as one portion of the vein bends away from its other portion (Fig. 18 (*c*)). In fractured folds, the ends of the faulted veins may bend in the same direction up or down, or in opposite directions.

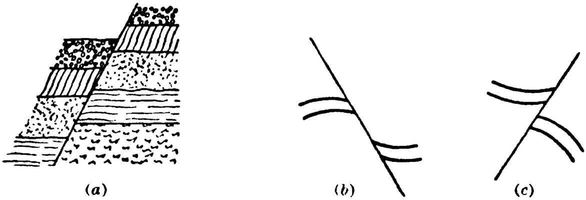

(*a*) (*b*) (*c*)

FIG. 18. DIRECTION OF FAULT. TOLD BY:
(*a*) Sequence of strata; (*b*) Bend of vein; (*c*) Larger angle

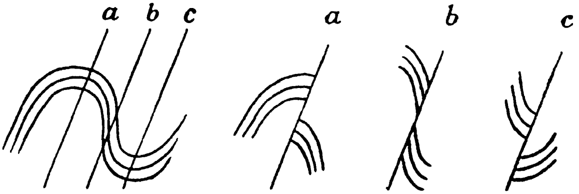


FIG. 19. FRACTURED FOLDS
(*a*) Anticline; (*b*) Limb; (*c*) Syncline

As to how the ends point in fractured folds largely depends on what part of the fold the fracture takes place, whether through an anticline or syncline of a leg (Fig. 19 (*a*) (*b*) (*c*)). Fractured folds are generally reversed faults, and the folded rocks assist one in solving the problem. Reverse fold faults are more frequent then true reversed faults and generally have a steeper plane of dislocation, besides dipping in the same direction as the strata.

To determine the throw of a faulted bed or vein in sedimentary strata, note the relative position of the strata if sufficiently distinctive in character.

To find the faulted part of a bed, follow the larger angle between the bed and the fault: that is, if the fault is first met with in the roof, seek up; if in the floor, seek down. This only holds good for normal faults; with reversed faults this is incorrect; with vertical faults it is indefinite. Fortunately there are more normal than reverse faults.

VALUABLE MINERAL DEPOSITS

Deposition of Ores

Igneous rocks may be divided into two main groups, acidic and basic, though there are others which are intermediate. The elements associated with acidic rocks such as granite, syenite and trachyte, are silicon, lithium, sodium, potassium, boron, fluorine, tellurium, cerium, yttrium, uranium, tin, thorium, tantalium, niobium, tungsten molybdenum, and beryllium. The elements which follow basic rocks such as diorite, andesite, gabbro and peridotite, are calcium, magnesium, iron, manganese, chromium, nickel, cobalt, sulphur, and often lead, zinc, copper, silver, gold, antimony, mercury, barium, strontium, titanium, selenium, and phosphorus. Most ore deposits are more or less associated with igneous rocks.

Two things are necessary for the deposition of ore: the source and a place where the ore can be deposited. The vehicle by which the minerals are conveyed from their source to the place where they are concentrated may be molten, as in the case of magmatic segregation; by vapour, or by heated mineralized water. The deposition may be due to a reduction of temperature and pressure as the vehicle nears the surface; or to the presence of a precipitant, such as carbonaceous shale, which in some places throws down gold. In some mines a vertical zonal change is found to take place in the primary ore due to original alterations in temperature and pressure.

Above water-level—which is not the same as sea-level, and varies with the undulations of the surface—there are also zonal variations, but due to another cause. The sulphides, which are stable under deep conditions, become unstable as they get near the surface, where they are oxidized and more or less soluble. Consider a silver-lead-zinc ore deposit. The actual outcrop may be leached. The silver does not travel far, as it is quickly thrown down by the chloride of sodium present. This is followed downward by carbonate of lead, and below this is sulphate of lead. Sulphate of zinc, being more soluble than the silver and lead salts, finds its way lower down. Of course, there is no sharp line between these zones which merge one into the other. As the water-level fluctuates, there is an intermediate zone where the primary minerals are altered. The zone of secondary enrichment often provides the means for opening up a mine and supplying the necessary plant. In other cases, this natural concentration is the only part of a lode that pays to mine. The depth to which the zone of secondary enrichment reaches depends on the climate and nature of the ground surface. In dry districts, it may reach down two or three hundred feet; in hilly country, where there is a great rainfall, the depth may be only a matter of an inch or two. The outcrop of present-day lodes is mostly stumps of the original deposits, and the minerals of the zone of secondary enrichment mostly came from the upper parts of the deposit long since weathered. On the surface joints of the rocks open out more than deeper down, and not infrequently secondary minerals, leached from the lode, occupy these joints and make the lode look wider on the surface than it really is, causing the miner much disappointment when he sinks.

In a molten magma, certain elements combine together before others. Thus we get magmatic segregations of iron ores and chromite. They are irregular in shape and have no definite walls.

Pegmatite or giant granite occurs in connection with granite intrusions and often contains as accessory minerals cassiterite, wolfram, and other valuable minerals besides books of mica.

Gases and vapour above their critical temperature may fill cavities with valuable minerals or cause replacements.

Earth strains may break up a strip of country without making a distinct fracture, to which underground waters find access and deposit ores. Such a deposit is known as a mullocky lode: it has no definite walls and its payable limits have to be determined by sampling and assaying.

Fissure veins are definite breaks in rocks due to earth strains. One wall moves on the other, and as the fracture is not regular, especially when passing from one class of rock to another, there is a series of comparatively wide and narrow places, both in length and depth. The wide places are known as "makes of ore" and the narrow parts as "pinches." It is not to be expected that a cavity of any size could remain open with fractured walls on either side. Pieces of wall rock broken off occupy these spaces and are either cemented together or are replaced by ores. When a large block of wall rock becomes detached, it is known as a "horse," and the vein on either side is said to "ride the horse." A fracture may take place suddenly, or may continue slowly for some considerable time, when the strain is more in the form of a tear. Though a rock strain may have been relieved for the time being, and any spaces in a fissure filled up with ore, if the strain is continued in the same direction, the original fracture being a line of weakness, it is more likely to open up again than to form a new fracture in undisturbed rock. The second fracture may be filled by the same class of ore as the first, or one of another kind depending on the source from which the thermal waters came. The boundary between the two fractures forms a wall between walls (Fig. 20), sometimes mistaken for an outside wall, which goes to show the advisability of occasionally putting a shot in either side to ascertain if there is any ore there or not. Signs of motion are shown by striæ on slickenslides if they exist. When a dig or gouge occurs on the wall of a vein, it serves a double purpose, inasmuch as it confines the ore solution to the fissure by preventing it from passing into the wall rock where it might be too poor to be worth while mining; secondly, it is easily picked out, and so leaves a face for ore to be "fired to," instead of shooting "in the solid."

When driving through a barren portion of a vein, miners with local knowledge can often tell when they are approaching a make of ore, if they meet with what is termed "kindly looking country," which is not so hard as the barren portion and may show signs of mineralization.

When ore occurs in the form of lenses, that part found above the widest portion will be found to be more extensive than that below where the rock is tighter. In many cases, pay ore occurs in shoots which pitch in the direction of the strike; but it is desirable to know which end of the vein the ore will pitch before deciding on the site for a shaft. This may be done before underground work has proceeded sufficiently far to supply the required information. If the vein cuts through sedimentary rocks, note the strike and underlie of the rock and also that of the vein. The direction of the line of intersection of the rock and vein is the direction of the pitch of the ore shoot (Fig. 21). If the country rock is of a massive nature, such as granite, in place of the strike of the sedimentary rock take the strike of the master joints. If a vein cuts through an anticline, the pitch of an ore shoot on one side will be in the opposite direction to the pitch on the other. In certain iron ore deposits, phosphorus is found to occur in shoots, and though this may have to be determined by careful sampling and assaying, when once found and kept separate from the ordinary run of ore, it may save much trouble and expense in the treatment of the ore later on.

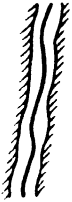


FIG. 20. WALL WITHIN WALLS

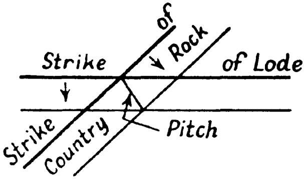


FIG. 21. TO FIND DIRECTION OF PITCH

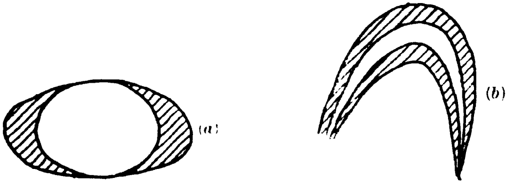


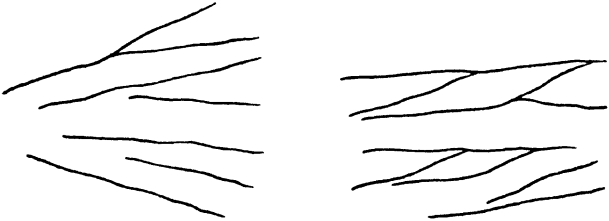
FIG. 22. SADDLE REEFS OR DOMES
(*a*) Plan. (*b*) Cross section

In some cases, ore only occurs in pipes at places where two veins cross which are otherwise barren. At such a crossing, earth strains appear to have broken up the rock and formed a passage for underground mineral bearing water.

So-called saddle reefs are really domes, the outcrop of which forms an oval. They occur in folded country and may sometimes be found one below another. The crest of the fold is often broken up owing to folding, and the legs gradually pinch out (Fig. 22 (*a*) (*b*)).

Veins may end by tapering out, or they may split up into veinlets. If a vein cuts out suddenly, one suspects a fault.

(*a*) Parallel



(*b*) Radiating (*c*) Linked

FIG. 23. VEINS

Some deposits have certain characteristics in common which enable them to be classified into types for convenience when describing them. Though we may speak of fissure veins, stockworks, impregnations, etc., as different types, and one or the other may predominate, we may find the three above mentioned in the same deposit.

Ore deposits generally occur in belts. These belts may extend for many miles in length, but any particular vein in it seldom extends over a mile. Where one vein exists, there are likely to be others. These may run approximately parallel to each other (Fig. 23 (*a*); radiate from a centre (Fig. 23 (*b*);

or be linked (Fig. 23 (*c*). When parallel, the veins get narrower and often poorer as they recede from the main lode. Linked veins run more or less parallel and are connected with counter veins, thus forming a network.

When the cap of a vein is broken up, but remains in the neighbourhood of the vein, it is known as an "elluvial" deposit. If the harder and tougher parts of the outcrop find their way into water courses, where they become waterworn, they are known as an "alluvial" deposit. Such alluvial deposits may contain gold, platinum metals, tin, wolfram, and various precious stones, not necessarily from the same source, though from the same watershed. Mineral veins mostly occur in the older rocks.

Stratified deposits are younger than their bed rock, but older than the rocks above them. Chemical and bacterial agents form certain bedded deposits in swamps, lakes, and seas, such as bog iron ore, clay, ironstone and bog manganese. In other cases, vegetable life may form deposits, such as coal, kerosene shale, and diatomaceous earth. In the case of coal, one bed is called a seam, but generally there are more than one that go to form coal measures.

Scale of Hardness

The hardness of a mineral is one of the first tests applied to it. The harder the mineral, the greater the noise it makes when rubbed with a file and the less powder is formed. Mohs' scale of hardness is that generally employed.

1. Talc, the common foliated variety, easily scratched with a thumb-nail.
2. Rock salt or gypsum.
3. Calcspar: transparent variety.
4. Fluorspar: crystallized variety.
5. Apatite: crystal.
6. Orthoclase felspar: cleavable variety.
7. Quartz crystal.
8. Topaz crystal.
9. Corrundum, sapphire.
10. Diamond.

Specific Gravity Solutions

These are used for determining the specific gravity of gems and other stones: they can be easily carried about in bottles.

Sonstedt's Solution. Double iodide of mercury and potassium. The density can be reduced by the addition of water. The solution is very poisonous.

Klein's Solution. Boro-tungstate of cadmium. This can also be diluted with water. It has the disadvantage of rapidly darkening on exposure to light, but the transparency can be restored.

Methylene Iodide. This is non-poisonous. It is light coloured, so mineral fragments can be readily seen. It can be clarified by shaking with a few drops of mercury, and can be diluted with benzine.

Of the above, methylene iodide is the most convenient. By means of it any density between 0·9 and 3·3 can be readily obtained, and it is possible to reach 3·6 by saturating it with iodoform.

Keep standard solutions in stoppered tubes, and obtain a series of minerals of known density for comparison.

LIST OF MINERALS, GIVING THEIR COMPOSITION, HARDNESS, SPECIFIC GRAVITY, CRYSTALLINE SYSTEM, AND COLOUR

C. = colourless, W. = white, R. = red, Bl. = blue, Y. = yellow, Ge. = green, Br. = brown, B. = black, V. = violet, Gr. = grey, Or. = orange.

Scale of Hardness. 1 = Talc, 2 = Gypsum, 3 = Calcite, 4 = Fluorspar, 5 = Apatite, 6 = Orthoclase, 7 = Quartz, 8 = Topaz, 9 = Corundum, 10 = Diamond.

Crystalline Systems. I Isometric, II Tetragonal, III Hexagonal, IV Orthorhombic, V Monoclinic, VI Triclinic, O Amorphous.

The inclined hemihedral forms for I–IV are indicated by ×, the parallel hemihedral forms by π.

Scale of Fusibility (v. Kobell). 1 Antimonite, 2 Natrolite, 3 Alamandite, 4 Actinolite, 5 Orthoclase (fuses only in small splinters in the hottest part of the blowpipe flame), 6 Bronzite (only the sharp edges of small splinters fuse round).

LIST OF MINERALS

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Actinolite	(Ca, Mg, Fe) SiO_3	5–6	3·02–3·16	V	Ge.
Adamite	$Zn_2(OH)AsO_4$	3·5	4·3	V?	Y. Bl. R. Ge. (moonstone)
Adularia (moonstone)	K_2O, Al_2O_3, $6SiO_2$	6–6·5	2·5–2·69	V	C.R.
Agate	SiO_2	7	2·5–2·8	O	R. Ge. Gr. Br. W.
Aikinite	$Pb_2Cu_2Bi_2S_6$	2–2·5	6·1–6·8	IV	Gr. B.
Alabandite	MnS	3·5–4	3·9–4	I	B. Br.
Alabaster	CaO, SO_3 + $2H_2O$	1·5–2	2·4	O	W.
Albite	Na_2O, Al_2O_3, $6SiO_2$	6–7	2·5–2·64	VI	C. Gr. Ge. Bl. R.
Allophane	Al_2SiO_5, $5H_2O$	3	1·8–1·9	O	Bl. Ge. Y. Br. C.
Almandine (garnet)	3FeO, Al_2O_3, $3SiO_2$	6·5–7·5	3–4·3	I	R. RBr.
Aluminite	Al_2O_3, SO_3, $9H_2O$	1–2	1·6	O	W.
Alunite	K_2O_3, Al_2O_3, $4SO_3$, 6aq.	2–2·5	1·7–2	Iπ	C. Y. Gr.
Amalgam	Ag + Hg	3–3·5	13·7–14·1	I	W. (silver)
Amazon st. (orthoclase)	K_2O, Al_2O_3, $6SiO_2$	6–6·5	2·4–2·6	V	Ge.
Amber (succinite)	Fossil resin	2–2·5	1·0	O	W. Y. Br. R.
Amblygonite	$2Al_2P_2O_8$ + 3Li (Na) F	6	3·05–3·1	VI	W. Gr. Ge.
Ambrite	C, O, H	2	1·034	—	Y. Gr. Ge.
Amethyst (quartz)	SiO_2	7	2·64–2·66	III	Bl. V.
Amphibole (hornblende)	SiO_3(Ca, Mg, Fe, Mn, Na_2, K_2, H_2)O	5–6	2·9–3·4	V	C. Ge. B.
Analcime (analcite)	Na_2O, Al_2O_3, $4SiO_2$, $2H_2O$	5·5–5	2·29	I	C. W. R. Gr. Ge. Y.
Anatase (octahedrite)	TiO_2	5·5–6	3·8–3·9	II	C. Br. B. Bl. Ge. Y.
Andalusite	Al_2O_3, SiO_2	7·5	3–3·2	IV	Gr. R. Br. W. V. Ge.
Andesine (andesite)	((CaNa$_2$)O, Al_2O_3, $4SiO_2$	5–6	2·65–2·74	VI	C. Y. Gr. W. Ge. R.
Andradite (garnet)	$Ca_3Fe_2Si_3O_{12}$	6·5–7·5	3·6–4	I	Y. Gr. R. Br. Ge. B.
Anglesite	PbO, SO_3	2·7–3	6–6·37	IV	C. Y. Ge. Gr. Br. W. Bl.
Anhydrite	CaO, SO_3	3–3·5	2·8–2·98	IV	C. W. Gr. Bl. R.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Annabergite	$3NiO, As_2O_5 + 8H_2O$	2–2·5	3–3·1	V	Ge. W.
Anorthite	$CaO, Al_2O_3, 2SiO_2$	6–7	2·6–2·78	VI	W. Gr. R. C.
Anthophyllite	$(Mg, Fe)O, SiO_2$	5·5	3·18–3·22	IV	Br. Gr. Ge. Y.
Anthracite	C (95%)	2–2·5	1·3–1·75	O	B.
Antimonite (stibnite)	Sb_2S_3	2	4·5	IV	Gr. B.
Antimony (native)	Sb (Ag, As, Fe)	3 3 5	6·6	III ×	W. Gr.
Apatite	$3Ca_3P_2O_8 + CaCl_2(CaF_2)$	4·5–5	2·9–3·2	IIIπ	C. Ge. Bl. Y. V. W. R. Gr. Br
Apophyllite	$4(H_2CaSi_2O_6 + aq) + KF$	4 5–5	2·3–2·4	II	C. R. Gr. Ge. Y.
Aquamarine	$Al_2O_3, 3BeO, 6SiO_2$	7·5–8	2·6–2·7	III	Bl. Ge.
Aragonite	$CaOCO_2$	3·5–4	2·95	IV	C. W. Y. Gr. Ge. V.
Argentite	Ag_2S	2–2·5	7·1–7·36	I	Gr.
Arkansite (brookite)	TiO_2	5.5.6	4	IV?	Br. Y. R. B. Gr.
Arsenic (native)	As (traces of Sb, Ag, Fe, Bi and Au)	3·5	5 9	III ×	W. Gr. B.
Arsenolite	As_2O_3	1·5	3·7	I	C. W. Y. R.
Asbestos (amianthus)	$(Mg, Ca)O, SiO_2$	5	3·02–3·1	V	W. Ge. Br.
Asbolite	$(Co, Cu)O$ $2MnO_2H_2O$	2.2 5	3·1–3·29	O	Br. B. Bl. B.
Asphaltum	C (76%) H, O.	1–2	1–1·7	O	Br. B.
Atacamite	$3CuO, CuCl_2, 3H_2O$	3–3·5	3·7–4	IV	Ge.
Augite (pyroxene)	$(Ca, Mg, Fe)O, SiO_2$	5–6	3·2 –3·5	V	B. Gr. Ge. Br.
Aurichalcite	$2CuO, 3ZnO, 2CO_2, 3H_2O$	2	—	?	Ge.
Autunite (uranite)	$CaU_2P_2O_{12}$ + 10aq.	2–2·3	3–3·6	IV	Y.
Aventurine	Partly quartz, partly felspar	—	—	—	Y. R. Br. Gr.
Axinite	$(CaO, Al_2O_3), SiO_2, Bo_2O_3$	6·5–7	3·3	VI	C. Br. V. Gr.
Azurite	$CuO, H_2O, 2(CuO, CO_2)$	3·5–4·2	3·5–3·8	V	Bl.
Babingtonite	$9(CaFeMn)SiO_3 \times Fe_2Si_3O_9$	5·5–6	3·4	VI	B. Ge.
Barytes (heavy spar)	BaO, SO_3	2·5–3·5	4·4–4·7	IV	C. Y. Gr. Bl. R. Br.
Barytocalcite	$BaOCO_2, CaOCO_2$	4	3·6	V	W. Gr. Y. Ge.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Basanite (touchstone)	SiO_2 impure with Fe	7	2·8	O	B.
Bauxite	$Al_2(Fe)_2O_3 + aq$	—	2·5	O	W. Br.
Beryl	$3BeO, Al_2O_3, 6SiO_2$	7·5–8	2·67–2·7	III	Ge. Y. Bl. W.
Bieberite(cobalt vitriol)	$CoO, SO_3 + 7H_2O$	—	1·9	V	R.
Biotote	$K_2SiO_4, (Fe[Mg])_2, SiO_4 + (Al[Fe])_2 Si_3O_{12}$	2·5–3	2·7–3·1	V	Ge. Br. B. Gr.
Bismite (bismuth ochre)	Bi_2O_3	Soft	4·3–4·7	?	Gr. Y.
Bismuth (native)	Bi (As, S, T)	2–2·5	9·7	III ×	W. R.
Bismuthinite	Bi_2S_3	2·0	6·4–7·2	IV	W. Gr. Y.
Bismutite	Bi_2O_3, CO_2	4–4·5	6·8–6·9	?	Gr. Ge. Y. W.
Blende (sphalerite)	ZnS	3·5–4	3·9–4·2	I ×	Y. Br. Ge. B. R.
Blödite	$Na_2O, MgO, 2SO_3, 4H_2O$	2·5–3·5	2·25	V	C. Ge. Gr. R.
Bole (halloysite)	Ferruginous clay	1–2	2–2·5	O	Br. Y. R.
Boracite	$Mg_7B_{16}O_{30}Cl_2$	7	2·97	I ×	C. Gr. Ge. Y.
Borax (tinkal)	$Na_2B_4O_7, 10H_2O$	2–2·5	1·71	V	C. W. Gr. Bl. Ge.
Bornite (erubescite)	$3Cu_2S, Fe_2S_3$	3	4·4–5·5	I	R. Br.
Boulangerite	$3PbS + Sb_2S_3$	2·5–3	5·7–6	O	Gr.
Bournotite	$Sb_2S_3, 2PbS, Cu_2S$	2·5–3	5·7–5·9	IV	Gr.
Braunite	Mn_2O_3	6–6·5	4·7–4·9	II	Br. B.
Bredbergite (garnet)	$(CaMg)_3Fe_2Si_3O_{12}$	6·5–7·5	3·2–4·3	I	—
Breithauptite	NiSb	5	7·5–7·6	III	R.
Bronzite (enstatite)	$(FeMg)O, SiO_2$	5·5	3·12–3·3	IV	Br. Y. Gr. Ge.
Brookite	TiO_2	5·5–6	4	IV?	Br. Y. R. B. Gr.
Brown coal (lignite)	C 55–75%	—	1·2–1·4	O	Br. B.
Brucite	MgO, H_2O	2·5	2·35	III ×	C. Ge. Gr. Bl.
Cacholong (opal)	$SiO_2 + 3–9\ aq$	5–6	2	O	W. Bl. Y. R.
Cacoxenite	$2Fe_2O_3, P_2O_5, 12H_2O$	—	2·3	V or VI	Y.
Cairngorm (quartz)	SiO_2	7	2·5–2·8	III	Y. Br. B.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Calamine	$2ZnO, SiO_2, H_2O$	4·5–5	3·1–3·9	IV	C. W. Ge. Y. R. Br. Gr. Bl.
Calcite	CaO, CO_2	2·5–3	2·723	III ×	C. Ge. Gr. Br. R. Bl. V. Y. B.
Caledonite	$5PbSO_4 + 3H_2CuO_2 + 2H_2PbO_2$	2·5–3	6·4	V	Ge. Bl.
Calomel	Hg_2Cl_2	1–2	6·5	II	W. Gr. Br.
Cancrinite (nepheline)	$(NaK)_2O, Al_2O_3, 2SiO_2$	5–6	2·45	III	Gr. Bl. R. Ge. Y. W.
Carnallite	$KCl, MgCl_2, 6H_2O$	—	1·618	IV	C. R.
Carnelian	SiO_2 with Fe_2O_3	7	2·65	O	R. Br.
Cassiterite	SnO_2(up to 9% Fe_2O_3)	6–7	6·4–7·1	II	Y. Br. Gr. R. B.
Cat's-eye	SiO_2	7	2·6	O	Gr.
Celestine	SrO, SO_3	3–3·5	3·96	IV	R. Bl. W.
Cerargyrite	$AgCl$	1–1·5	5·6	I	Ge. Gr. V. W.
Cerite	$2(Ce, La, Di)O, SiO_2, H_2O$	5·5	4·9–5	IV	Br. R.
Cerussite	PbO, CO_2	3–3·5	6·4	IV	C. W. Gr. B. Br.
Cervantite	SbO_2	4–5	4·08	IV	Y.
Chabazite	$CaO, Al_2O_3, 4SiO_2, 6H_2O$	4–5	2	III ∧	C. R.
Chalcanthite	$CuO, SO_3.5H_2O$	2·5	2·2	VI	Bl.
Chalcedony (quartz)	SiO_2	7	2·65	O	W. Gr. B. Y. Ge. R. Bl. Br.
Chalcopyrite (copper pyrites)	Cu_2S, Fe_2S_3	3·5–4	4·1–4·3	II >	Y.
Chalcocite (copper glance)	Cu_2S	2·5–3	5·5–5·8	IV	Gr.
Chalcostibnite	Cu_2S, Sb_2S_3	3·5	4·7	IV	Gr. B.
Chalybite (siderite)	FeO, CO_2	3·5–4·5	3·7–3·9	III ∧	C. Gr. R. Y. B.
Chiastolite (andalusite)	Al_2O_3, SiO_2	3–2·5	2·9–3	IV	W. Gr. Y. R.
Chlorite	$8MgO, Al_2O_3, 5SiO_2, 7H_2O$	1–3	2·8	III	Ge.
Chondrodite	$5MgO, 2SiO_2$, or $4MgO, MgFl_2, 2SiO_2$	6–6·5	3·17–3·23	V	Y. R. Br. Ge. Gr. B.
Chromite	$FeCr_2O_4$	5·5	4·3–4·5	I	B.
Chrysoberyl	BeO, Al_2O_3	8·5	3·65–3·8	IV	Ge. R.
Chrysocolla	$CuO, SiO_2, 2H_2O$	2–4	2–2·3	O	Ge. Br. Bl. B.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Chrysolite (olivine or peridot)	$2(MgFe)O, SiO_2$	6–7	3·1–3·5	IV	Y. Gr. Ge. Br.
Chrysoprase (chalcedony)	SiO_2 . . .	7	2·65	O	Ge.
Cinnabar .	HgS . . .	2–2·5	8·99	III	R. Gr. Br.
Clausthalite .	$PbSe$ . . .	2·5–3	7 6–8·8	I	Gr.
Clinochlore (ripidolite)	$5Mg(Fe)O, Al_2O_3, 3SiO_2 + 4H_2O$	2–2·5	2·65–2·78	V	Ge. B. Bl. R.
Clinoclasite .	$6CuO, As_2O_5, 3H_2O$	2 5–3	4·1–4·4	V	Ge. Bl.
Coal (mineral)	$C(+O+H+N)$, 74–96% C.	0 5–2·5	1–1·8	O	B.
Cobaltite (glance cobalt)	$CoS_2 + CoAs_2$.	5·5	6–6·3	Iπ	R. W. Gr.
Collyrite .	$2Al_2O_3, SiO_2, 8H_2O$	1·2	2	O	W.
Columbite (niobite)	$FeO(Nb,Ta)_2O_5$	6	5·4–6·4	IV	Br. Gr. B.
Copiapite .	$2Fe_2O_3, 5SO_3, 12H_2O$	1·5	2·14	III	Y.
Copper (native)	Cu	2·5–3	8·8	I	R.
Copper glance	Cu_2S . . .	2·5–3	5·5–5·8	IV	Gr.
Copper nickel (niccolite)	$NiAs$ . . .	5–5·5	7·33–7·6	III	R. Gr.
Copper pyrites (chalcopyrite)	Cu_2S, Fe_2S_3 .	3·5–4	4·1–4·3	II \	Y.
Copper vitriol (chalcanthite)	$CuO, SO_3, 5H_2O$	2·5	2·2	VI	Bl.
Coquimbite .	$(FeAl)_2O_3, 3SO_3, 9H_2O$	2–2·5	2–2·1	III	C. W. Bl. Ge. V.
Cordierite (iolite)	$3MgO, 3Al_2O_3, 8SiO_2$	7–7·5	2·6–2·7	IV	W. Br. Y. Gr. Bl.
Corundum (emerald, sapphire, ruby)	Al_2O_3 . . .	9	3·9–4·2	III ×	C. W. Gr. Y. Br. Bl. R. Ge.
Cotunnite .	$PbCl_2$ . . .	2	5·2	IV	W.
Covellite (indigo copper)	CuS . . .	1·5–2	3·8–3·9	III	Bl. B.
Crocoisite .	PbO, CrO_3 .	2·5–3	5·9–6·1	V	R.
Cryolite .	$Al_2Fl_6, 6NaFl$.	2·5	2·9–3	VI	W. Gr. B. R. Br.
Cuprite .	Cu_2O . . .	3·5–4	5·7–6·1	I	R.
Cyanite (disthene)	Al_2O_3, SiO_2 .	5–7	3·4–3·68	VI	B. Bl. W. Gr. Ge.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Danburite .	$CaO, B_2O_3, 2SiO_2$	7	2·95	IV	Y.
Datolite .	$2CaO, 2SiO_2, B_2O_3, H_2O$	5–5·5	2·8–3	V	W. Gr. Ge. Y. R. V.
Descloizite .	$4(Pb, Zn)O, V_2O_5, H_2O$	3·5	5·8	IV	Ge. B.
Desmine (stilbite)	$CaO, Al_2O_3, 6SiO_2, 6H_2O$	3·5–4	2·1	IV	W. C. Y. Br. G.
Diallage .	$(Ca, Mg, Fe)O, SiO_2$	4	3·2–3·3	V	Ge. Br.
Dialogite . (rhodochrosiderite)	$Mn(Ca)O, CO_2$	3·5–4·5	3·4–3·7	III	R. Y. Br.
Diamond .	C . . .	10	3·5	I	C. R. Y. Bl. B. Br. Ge
Diaspore .	Al_2O_3, H_2O .	6·5–7	3·3–3·5	IV	Gr. Ge. Y. R. W. Br.
Dichroite (cordierite)	$3MgO, 3Al_2O_3, 8SiO_2$	7–7·5	2·6–2·7	IV	C. W. Br. Y. Gr. Bl.
Diopside (pyroxene)	$(Ca, Mg)O, SiO_2$	5–6	3·2–3·38	V	C. Y. Gr. Ge.
Dioptase .	CuO, SiO_2, H_2O	5	3·27–3·348	III	Ge.
Disthene (cyanite)	Al_2O_3, SiO_2 .	5–7·2	3·4–3·68	VI	Ge. Bl. W. Gr. B.
Dolomite .	$CaO, MgO, 2CO_2$	3·5–4	2·8–2·9	III	C. Gr. Ge. R. Br. B.
Dufrenite .	$2Fe_2O_3, P_2O_5, 3H_2O$	3·5–4	3·3–3·5	IV	Ge. Y. Br.
Dufrenoysite	$2PbS, As_2S_3$.	3	5·57	IV	Gr.
Durangite .	$Al_2O_3, As_2O_5, 2NaFl$	5	3·9–4	V	R. Y.
Dyscrasite .	Ag_4Sb or Ag_6Sb	3·5–4	9·4–9·8	IV	W. Gr.
Elaterite .	C_nH_{2n} . . .	Soft	0·8–1·23	O	B. Br. R. Y.
Electrum .	Au + Ag . .	2·5–3	13–15·5	I	Y. W.
Embolite .	Ag(Cl, Br) . .	1–1·5	5·3–5·4	I	Gr. Ge.
Emerald .	Al_2O_3 . . .	9	3·9–4·2	III	Ge.
Emplectite .	Cu_2S, Bi_2S_3 .	2	5·1–5·26	IV	W. Gr.
Enargite .	Cu_3AsS_4 . .	3	4·4	IV	Gr. B.
Enstatite .	MgO, SiO_2 . .	5·5	3·1–3·3	IV	C. W. Ge. Y. Gr. Br.
Epidote .	$3Al_2O_3, 4CaO, 6SiO_2, H_2O$	6–7	3·5	V	Gr. Br. B. R. Y. Ge.
Epistilbite .	$CaO, Al_2O_3, 6SiO_2, 5H_2O$	4–5	2·2–2·36	IV	W. Bl.
Epsomite .	$MgSO_4, 7H_2O$.	2–2·5	1·7	IV	C. W. Gr.
Erubescite (purple copper ore)	$3Cu_2S, Fe_2S_3$	3	4·4–5·5	I	R. Br.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Erythrite (cobalt bloom)	3CoO, As_2O_5, $8H_2O$	1·5–2·5	3	V	R. Gr.
Euchroite .	4CuO, As_2O_5, $7H_2O$	3·5–4	3·4	IV	Ge.
Euclase	2BeO, Al_2O_3, $2SiO_2$, H_2O	7·5	3·1	V	C. Gr. B.
Eudialyte .	$2Na_2O_{12}$, CaO, $6(SiZn)O_2$?	5–5·5	2·8–3	III ×	R. Br.
Eulytite .	$2Bi_2O_3$, $3SiO_2$	4·5–5	6·106	I	Br. Gr. Y. W.
Euxenite .	$[(YCaCeUTh)(Cb. TaTi)_2O_6]$	5·25–6·5	4·6–4·99	IV	B. Gr.
Fahlerz . . (grey copper ore)	$4Cu_2$ (Fe, Zn, Pb, Hg, Ag_2) S + Sb_2 (As, Bi)S_3	3–4·5	4·5–5·5	I	Gr. B.
Fahlunite .	Variable . .	3·5–5	2·6–2·8	?	Gr. Ge. Br. B.
Fassaite . (pyroxene augite)	$RSiO_3$,(R = Ca, Mg, Fe, Mn)	6	3·25–3·5	V	Ge. B.
Fayalite (olivine)	2Fe(Mn, Ca, Mg)O, SiO_2	6	4–4·15	IV	Ge. Br. B.
Felspar (anorthite, albite, andesite, labradorite microdine, oligoclase, orthoclase)	Al_2O_3, $3SiO_2$ (K. Na)				
Fergusonite .	R_2Q_2O (R = Y, Ce, W, Fe, Ca; Q = Nb, Ta)	5·5–6	5·8	II	B. Br.
Fibrolite (sillimanite)	Al_2O_3, SiO_2 .	6-7	3·23	V	Gr. Br. Ge.
Fichtelite .	C_nH_{2n} . . .	1	—	V	Br. W.
Flint(quartz)	SiO_2 . . .	7	2·63	O	Gr. Br. B. Bl. R. Y.
Fluorspar (fluorite)	$CaFl_2$ . . .	4	3·18	I	C. V. Ge. Bl. Y. R.
Fowlerite .	(Mn, Fe, Zn, Mg, Ca))O, SiO_2	4–5	3·4	VI	Br. R. Y.
Franklinite .	(Fe, Zn, Mn), Fe_2O_4	5·5–6·5	5·06	I	R.
Freibergite .	$(Ag, Cu)_8$, Sb_2S_7, + $(Fe, Zn)_4$, Sb_2S_7	3–4	4·8–5	I ×	Gr. B.
Fuchsite .	—	2-2·5	2·75	V	Ge.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Gadolinite	$3(Y, La, Fe, Be)O, SiO_2$	6·5–7	4·2–4·35	V	B. Ge.
Gahnite	ZnO, Al_2O_3	7·5–8	4–4·6	I	Gr. Bl. B. Ge. Y.
Galena	PbS	2·5–2·7	7·2–7·7	I	Gr.
Garnet (almandite, melanite, grossularite, etc.)	$3RO, R_2O_3, 3SiO_2, R = Ca, Fe, Mg, Mn; R_2 = Al_2, F_2, Cr_2$	6·5–7·5	3·1–4·3	I	B. Ge. Y. R. Br. W.
Gaylussite	$Na_2O, CaO, 2CO_2, 5H_2O$	2–3	1·99	V	C. Y.
Genthite	$H_4(Ni, Mg)_4Si_3O_{12}$	3–4	2·4	O	Ge. Y.
Gersdorffite	$NiS_2 + NiAs_2$	5·5	5·6–6·9	I	W. Gr.
Geyserite (opal)	$SiO_2 + Aq.$	5	2	O	Gr. W.
Gibbsite	$Al_2O_3, 3H_2O$	2·5–3·5	2·35	V	C. W. Gr. Ge. R.
Gismondite	$CaO, Al_2O_3, 2SiO_2 + 4H_2O$	5	2·26	II	Gr. R.
Glauberite	$Na_2O, CaO, 2SO_3$	2·5–3	2·64–2·85	V	C. R. Y. Gr.
Glaubersalt (mirabilite)	$Na_2O, SO_3, 10H_2O$	1·5–2	1·4–1·5	V	C.
Gold	$Au(Ag, Cu, Fe, Pb)$	2·5–3	15·6–19·5	I	Y.
Goslarite	$ZnO, SO_3 + 7H_2O$	2–2·5	1·9–2·1	IV	W. C. R. Bl.
Göthite	Fe_2O_3, H_2O	5–5·5	4–4·4	IV	Br. Y. R.
Graphite (plumbago)	C. often associated with SiO_2, CaO, Fe_2O_3	0·5–2	2·1–2·2	III	Gr. B.
Greenockite	CdS	3–3·5	4·8–4·9	III	Y. O.
Grossularite (garnet)	$Ca_3Al_2Si_3O_{12}$	6·5–7·5	3·1–3·7	I	Ge. Br. W. Y.
Gypsum (selenite)	$CaO, SO_3, 2H_2O$	1·5–2	2·33	V	C. Y. Br. W. R. Gr. Bl.
Haidingerite	$2CaO, As_2O_5, 2H_2O$	2–2·5	2·8–2·9	IV	C. W.
Halite (rock salt)	$NaCl$	2–2·5	2·25	I	Gr. R. Bl. W. Y.
Halloysite	$Al_2O_3, 2SiO_2, 4H_2O$	1·5–2·5	1·9–2·1	O	W. Bl. Y. Gr. Ge.
Harmotome	$BaO, Al_2O_3, 5SiO_2, 5H_2O$	4·5	2·45	V	C. W. Br. Gr. R. Y.
Haucrite	MnS_2	4	3·46	Iπ	Br. B.
Hausmannite	Mn_3O_4	5–5·5	4·7–4·9	II	Br. B.
Haüynite	$2Na_2(Ca)Al_2Si_2O_8 + CaSO_4$	5·5–6	2·4–2·5	I	Bl. Gr.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Heavy spar (barytes)	BaO, SO_3	2·5–3·5	4·3–4·7	IV	C. Y. Gr. Bl. R. Br.
Heliotrope (bloodstone)	Chalcedony	7	2·65	O	Ge. with R. spots
Helvite	$(Be, Mn, Fe)O, SiO_2, MnS$	6–6·5	3·1–3·3	I ×	Y. Br. Ge.
Hæmatite (specular iron)	Fe_2O_3	5·5–6·5	4·5–5·28	III ×	Gr. R. B.
Hessite	Ag_2Te	2–3·5	8–8·6	IV	Gr.
Heulandite (stilbite)	$CaO, Al_2O_3, 6SiO_2, 5H_2O$	3·5–4	2·2	V	C. R. Gr. Br.
Hornblende (amphibole)	$(Ca, Mg, Fe)O, SiO_2$	5–6	2·9–3·4	V	Ge. B. C.
Hornsilver (cerargyrite)	$AgCl$	1–1·5	5·5	I	Ge. Gr. W. V.
Hornstone (chert)	SiO_2	7	2·65	Compact	Ge. Y. Gr. R. Br.
Huautajayite	$(Na, Ag)Cl$	—	—	I	W.
Hübnerite	MnO, WO_3	4·5	7·14	V	Br.
Humite	$4MgO, MgFl_2, SiO_2$	6–6·5	3·01–3·23	IV	Y. R. Br. W.
Hyacinth (zircon)	$ZrO_2 + SiO_2$	7·5	4–4·7	II	B. R. O.
Hyalite (opal)	$SiO_2 + Aq.$	6	2·15–2·18	O	C.
Hyalophane	$K_2O, Al_2O_3, 6SiO_2 + BaO, Al_2O_3, 2SiO_2$	6–6·5	2·8	V	W. C. R.
Hyalosiderite (olivine)	$4MgO, 2FeO, 3SiO_2$	6–6·5	3·5	IV	Ge. Y. Br. R.
Hydromagnesite	$4MgO, 3CO_2, 4H_2O$	3·5	2·15	V	W.
Hydrozincite	$ZnOCO_2, 2ZnO, H_2O$	2–2·5	3·6–3·8	?	W. Gr. Y.
Hypersthene	$(Mg, Fe)O, SiO_2$	5–6	3·4	IV	Br. Gr. Ge. B.
Idocrase (vesuvianite)	$2(Al_2Ca_3)O_3, 3SiO_2$	6·5	3·3–3·45	II	Ge. Br. Bl. Y.
Idrialite	C_nH_{2n-18}	1–1·5	1·4–1·6	O	Br. B. R.
Ilmenite (titanic iron)	FeO, TiO_2	5–6	4·5–5	III ×	B. Br. R.
Ilvaite	$H_2Ca_2Fe_5Si_4O_{18}$	5·5–6	3·7–4·2	IV	B.
Iodargyrite	AgI	Soft	5·5–5·7	I	Y.
Iolite	$3MgO, 3Al_2O_3, 8SiO_2$	7–7·5	2·6–2·7	IV	W. Br. Y. Gr. Bl.
Iridium	Ir + Os (Rh, Ru, Pt, Fe, Pd)	6–7	19–21·2	III ×	W. Gr.
Iridosmine	Ir + Os (Rh, Ru, Pt, Fe, Pd)	6–7	19–21·12	III ×	W. Gr.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Jamesonite .	$2PbS, Sb_2S_3$.	2–3	5·5–5·8	IV	Gr.
Jasper (quartz)	$SiO_2(Fe_2O_3)$.	7	2·65	O	R. Y. Br. Ge. B.
Jeffersonite .	$Ca(Fe, Mn, Zn, Mg)O, SiO_2$	4·5	3·5	VI	Ge. Br. B.
Jet . . .	C.H.O.	—	1·2–1·4	O	B.
Kainite . .	$KCl, MgO, SO_3. 3H_2O$	2·5	2·13–2·2	V	C. Gr. Y.
Kaluszite (syngenite)	$K_2O,SO_3 + CaO SO_3 + H_2O$	2·5	2·6	V	C.
Kaolinite .	$Al_2O_3, 2SiO_2, 2H_2O$	1–2·5	2·4–2·6	IV	W. Gr. Y. Br. Bl. R.
Kermesite .	Sb_2S_2O .	1–1·5	4·5	V	R.
Kieserite . .	MgO, SO_3, H_2O	3	2·51–2·57	V	C. W. Gr. Y.
Labradorite .	$(NaCa)O,Al_2O_3, 3SiO_2$	6	2·67–2·76	VI	C. W. Br. Gr. Ge. Bl
Lanarkite .	PbO,SO_3,PbO_2	2–2·5	6·3–6·4	V	Ge. Y.
Lanthanite .	$LaO, CO_2,3H_2O$	2·5–3	2·6	IV	W. Gr. Y.
Lapis lazuli	Varies, $Al_2O_3 + Na_2O + CaO + SO_3 + SiO_2$	5·5	2·4	I	Bl.
Lazulite .	$(Mg, Fe)O, Al_2O_3,P_2O_5, H_2O$	5–6	3·1	V	Bl.
Leadhillite .	$PbO, SO_3, 2(PbO, CO_2), PbO, H_2O$	2·5	6·26–6·4	IV	Y. W. Ge. Gr. Br.
Lepidolite .	(Al, K, Li), Fl, SiO_2	2·5–4	2·84–3	IV	R. V. Y. W.
Lepidomelane	$2(Fe, Mg, K_2)O OAl_2O_3, SiO_2$	3	3	IV	Ge. B.
Leucite . .	$K_2O, Al_2O_3, 4SiO_2$	5·5–6	2·45–2·5	II	C. Gr. W.
Leucophanite	$5(CaBe)O, 5SiO_2$, 2NaFl	3·5–4	2·97	IV	Ge. Br. Y.
Leucopyrite .	Fe_2As_3 . . .	5–5·5	6·8–8·7	IV	W. Gr.
Libethenite .	$4CuO, P_2O_5, H_2O$	4	3·6–3·8	IV	Ge.
Lievrite (ilvaite)	$H_2Ca_2Fe_5Si_4O_{18}$	5·5–6	3·7–4·2	IV	B. Br. Ge.
Lignite . .	60–70% C.; H, O, and ash	—	1·2–1·3	O	Br. B.
Limonite .	$2Fe_2O_3, 3H_2O$.	5–5·5	3·6–4	O	Br. Y. B.
Linarite . .	$(PbCu)O, SO_3 + (PbCu)O, H_2O$	2·5	5·3–5·45	V	Bl.
Liroconite .	$Cu_3(Al_2)As_2(P_2)O_8 + H_6(Cu_3, Al_2)O_6 + 9Aq$	2–2·5	2·8–2·9	V	Bl. Gr.
Lithomarge	Clay . . .	2–2·5	2·3–2·6	O	W. Y. Gr. R.
Löweite . .	$2(MgOSO_3,Na_2OSO_3), 5H_2O$	2·5–3	2·37	II	W. R. Y.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Lydian stone	SiO_2 impure with Fe, etc.	7	2·8	O	B.
Magnesite	MgO, CO_2	3·5–4·5	2·9–3·1	III ×	W. Y. Br.
Magnetite	Fe_3O_4	5·5–6·5	4·9–5·2	I	B.
Malachite	$2CuO, CO_2, H_2O$	3·5–4	3·7–4	V	Ge.
Manganite	Mn_2O_3, H_2O	4	4·2–4·4	IV	Gr. B.
Marble (calcite)	CaO, CO_2	2·5–3·5	2·5–2·8	—	W. Gr. Bl. Ge. Y. R. B.
Marcasite	FeS_2	6–6·5	4·6–4·8	IV	Y. Gr.
Margarite	$CaO, 2Al_2O_3, SiO_2, H_2O$	3·5–4·5	2·9–3	—	W. Gr. Y. R.
Mascagnite	$(NH_4O)_2SO_3, H_2O$	2–2·5	1·7–1·8	IV	C. W. Y.
Meerschaum (sepiolite)	$2MgO, 3SiO_2 + 2aq$	2–2·5	0·98–1·2	O	W. Gr. Y. R.
Meionite	$6CaO, 4Al_2O_3, 9SiO_2$	5·5–6	2·6–2·74	II	C. W.
Malenite (garnet)	$Ca_3Fe_2Si_3O_{12}$	6·5–7	3·5	I	B.
Melanterite (iron vitriol)	$FeO, SO_3 + 7H_2O$	2	1·8–1·9	V	Ge. Y.
Melilite	$2Al_2(Fe_2)O_3, 12CaO, Mg, 9SiO_2$	5–6	2·9	II	C. W. Y. Br.
Mellite	$C_6(CO_2)_6Al_2O_3, 18H_2O$	2–2·5	1·55–1·6	II	Y. R. Br. W.
Menaccanite (ilmenite)	FeO, TiO_2	5–6	4·5–5	III ×	B. Br.
Mendipite	$Pb_3O_2Cl_2 = (2PbO, PbCl_2)$	2·5–3	7–7·1	IV	W. Y.
Mesitite	$2MgO, CO_2 + FeO, CO_2$	4–5	3·3	III ×	W. Br. Y.
Mesolite	$CaO, Na_2O, 2Al_2O_3, 6SiO_2, 5H_2O$	5	2·2–2·4	V or VI	C. W. Y.
Mesotype (natrolite)	$Al_2O_3, Na_2O, 3SiO_2, 2H_2O$	5–5·5	2·17–2·2	IV	C. W. Y. Gr. R.
Miargyrite	$AgSbS_2$	2–2·5	5·2–5·4	V	Gr. B.
Microcline	$K_2O, Al_2O_3, 6SiO_2$	6	2·54	VI	W. R. Br.
Millerite	NiS	3–3·5	4·6–5·6	III ×	Y.
Mimetite	$3(3PbO, As_2O_5) PbCl_2$	3·5	7·2–7·25	III	C. Y. Or. Br.
Minium	Pb_3O_4	2–3	4·6	O	R. Y.
Mirabilite (glauber salt)	$Na_2O, SO_3, 10H_2O$	1·5–2	1·4–1·5	V	C.
Mispickel (arsenical pyrites)	$FeS_2 + FeAs_2$	5·5–6	6–6·4	IV	W. Gr.
Molybdenite	MoS_2	1–1·5	4·4–4·8	III?	Gr.
Molybdite	MoO_3	1–2	4·5	IV	Y.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Monazite .	$(Ce, La, Di, Th)_3, (PO_4)_2$	5–5·5	4·9–5·2	V	Br. R. Y.
Monticellite.	$2(Ca,Mg, Fe)O, SiO_2$	5–5·5	3·1	V	C. Gr. Ge.
Morenosite . (nickel vitriol)	$NiOSO_3+7H_2O$	2–2·25	2	IV	Ge.
Mosandrite .	$3CaO, 2(Ce, La, Di)_2O_3 5(SiO_2, TiO_2)$	4	3	IV	Br. R. Y.
Nagyagite .	$PbAu(TeS)_2$.	1–1·5	6·8–7·2	II	Gr.
Naphtha (rock oil)	C_nH_{2n+2} . .	—	0·7	O	C.
Natrolite (mesotype)	$Al_2O_3, Na_2O, 3SiO_2, 2H_2O$	5–5·5	2·17–2·2	IV	C. W. Y. Gr. R.
Naumannite	Ag_2Se . . .	2·5	8	I	B.
Nephelite .	$4Na_2O, 4Al_2O_3, 9SiO_2$	5·5–6	2·56–2·6	III ×	Gr. C. W. Y. Ge. Br R. .
Nephrite (jade)	$(Ca, Mg, Fe)O, Al_2O_3, SiO_2$	6–6·5	2·9–3·1	V	W. Gr. Ge.
Niccolite . (copper nickel)	$NiAs$ . . .	5–5·5	7·33–7·6	III	R .Gr.
Niobite (columbite)	$FeO(Nb,Ta)_2O_5$	6	5 4–6·4	IV	Br. B. Gr.
Nitre . .	Na_2O, N_2O_5 .	1·5–2	1 9–2	IV	C. Gr. Y. R.
Nosean (haüyne)	$Na_2O, Al_2O_3, 2SiO_2+ Na_2O, SO_3$	5·5	2·28–2·4	I	Ge. Bl.
Octahedrite (anatase)	TiO_2 . . .	5·5–6	3·8–3·9	II	C. Br. B. Bl. Ge. Y.
Oligoclase .	$2(Na_2Ca)O, 2Al_2O_3, 9SiO_2$	6–7	2·56–2·73	VI	C. Ge. R. Gr.
Olivene (chrysolite)	$2MgO, SiO_2$.	6–7	3–3·3·5	IV	Gr. Ge. Y. Br. R.
Olivenite .	$4CuO, As_2O_5, H_2O$	3	4·1–4·4	IV	Ge. Y. Br.
Onyx . .	SiO_2 . . .	7	2·65	O	Gr. W. B. Br.
Opal (precious, fire, common, wood, etc.)	$SiO_2 10H_2O$.	5·5–6·5	1·9–2·3	O	C. Y. Gr. R. Br. Ge.
Orpiment .	As_2S_3 . . .	1·5–2	3·48	IV	Y.
Orthite . .	$(Al_2,Fe_2,Ce_2)O_3, 2(CaFe)O, 3SiO_2$	5–6	3·1–4	V	B.
Orthoclase (felspar)	$K_2O, Al_2O_3, 6SiO_2$	6–6·5	2·4–2·7	V	C. R. Gr. Ge.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Ouvaroite	$Ca_3Cr_2Si_3O_{12}$	7·5	3·4–3·5	I	Ge.
Ozocerite	C_nH_{n2}	1	0·85–0·9	O	W. Y. Br. B. C. Ge.
Palladium (native)	Pd(PtIr)	4·5–5	11·3–11·8	I	Gr.
Paragonite	Na_2O, Al_2O_3, $2SiO_2$	2·5–3	2·78	O	W. Gr. Y. Ge.
Pectolite	Na_2O, 4CaO, $6SiO_2$ + Aq.	5	2·8	V	W. Gr.
Pentlandite	(NiFe)S	3·5–4	4·6	IV	Y.
Periclase	MgO	6	3·67	I	Gr.
Pericline (albite)	Na_2O, Al_2O_3, $6SiO_2$	6–6·5	2·6–2·64	VI	W.
Peridot (chrysolite, olivene)	2MgO, SiO_2	6–7	3·3–3·5	IV	Gr. Ge. Y. Br.
Perofskite	(Ca, Fe)O, TiO_2	5·5	4	I	R. Y. B.
Petalite	Al_2O_3, $Li_2(Na_2$, Ca)O, SiO_2	6–6·5	2·4–2·5	V	W. R. Gr. Ge.
Petroleum (mineral oil)	C_nH_{2n+2}	—	0·7–0·9	O	C. Y. Br.
Pharmacolite	2CaO, As_2O_3, $H_2O + 5H_2O$	2–2·5	2·6–2·7	V	W. Gr. R.
Pharmacosiderite	$3Fe_2As_2O_3 + H_6Fe_2O_6$	2·5	2·9–3	I ×	Ge. Br. Y. R.
Phenacite	2BeO, SiO_2	7·5–8	3	III	C. Y.
Phillipsite	CaO, Al_2O_3, $4SiO_2$, $4H_2O$	4–4·5	2·2	IV	C. W. R.
Phosgenite	PbO, CO_2, $PbCl_2$	2·5–3	6–6·3	II	W. Y. Gr.
Phosphorite (earthy apatite)	$3Ca_3P_2O_8 + CaF(Cl)_2$	5	3·15	O	W. Gr. Y.
Piauzite	Fossil resin	1·5	1·2	O	Br. B. Ge.
Pimelite	$(Al, Ni)_2O_3$, MgO, SiO_2	2·5	2·2–2·7	O	Ge.
Pisanite (iron copper vitriol)	(FeO, CuO)SO_3 + 7Aq.	2·5	2·2	VI	Bl.
Pistazite (epidote)	$3Al_2O_3$, 4CaO, $6SiO_2 + H_2O$	6–7	3·3–3·5	V	Ge. Y. B.
Pitchblende (uraninite)	U_2O_4(Pb, Fe, Ag, Ca, Mg, Bi, SiO_2, etc.)	5·5	6·4	I	B.
Plagionite	4PbS, $3Sb_2S_3$	2·5	5·4	V	Gr.
Platinum (native)	Pt(Fe, Ir, Rh, Pd, Os, Cu)	4–4·5	16–19	I	Gr.
Platin-iridium	Pt + Ir(Rh, Fe, Cu, Pd)	6–7	22·6–23	I	W. Gr.
Plattnerite	PtO_2	5–5·5	8·5	III	B.
Pleonaste (spinel)	MgO, Al_2O_3	8	3·5–4·1	I	Ge. Br. B. R. Bl. Y.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Polianite	MnO_2	6·5–7	4·8–5	IV	Gr.
Pollucite (pollux)	Cs_2O, Al_2O_3, $5SiO_2$, H_2O	5·5–6·5	2·9	I	C.
Polybasite	$(Sb, As)_2S_3$, $9(Ag_2, Cu_2)S$	2–3	6·21	IV	B.
Polycrase	$4RTiO_3 + RNb_2O_6 + 2H_2O$[R = Y, Er, Ce, (U, Fe)]	5·5	5–5·1	IV	B. Br.
Polyhalite	(Ca, K_2, Mg)O, SO_3, $2H_2O$	2·5–3	2·77	V ?	R. Y.
Prase	SiO_2	7	2·62	O	Ge.
Praseolite	Decomposed cordierite	3·5	2·75	IV	Ge.
Prehnite	Al_2O_3, 2CaO, $3SiO_2$, H_2O	6–6·5	2·8–2·9	IV	Ge. Gr. W.
Proustite	$3Ag_2S + As_2S_3$	2·2·5	5·4–5·56	III ×	R.
Psilomelane	$RO + 4MnO_2$ [R = K_2,Mn or Ba]	5–6	3·7–4·7	O	B. Gr.
Pucherite	Bi_2O_3, V_2O_5	6	6·25	IV	R. Br.
Pyrargyrite	Ag_3SbS_3	2·2·5	5·7–5·9	III ×	R. Gr.
Pyrites (iron pyrites)	FeS_2	6–6·5	4·8–5·2	Iπ	Y.
Pyrochlore	$Nb_2O_5 + TiO_2 + ThO_2 + CaO + CeO + FeO + HgO + Na_2O + H_2O$	5–5·5	4·3	I	Br. R.
Pyrolusite	MnO_2	2–2·5	4·82	IV	B. Gr.
Pyromorphite	$3(PbO)_3P_2O_5 + PbCl_2$	3·5–4	6·5–7	IIIπ	Ge. Br. Y. Gr. W.
Pyrope (garnet)	(Mg, Ca, Fe, $Mn)_3Al_2Si_3O_{12}$	6·5–7·5	3·5–4	I	R.
Pyrophillite (agalmatolite)	Al_2O_3, $3SiO_2$ + Aq.	1–2	2·8–2·9	IV	W. Gr. Ge. Br. Y.
Pyrrhotite (magnetic pyrites)	Fe_nS_{n+1} (generally Fe_7S_8)	3·5–4·5	4·4–4·6	III	Br. Y. R.
Pyroxene (augite, diallage diopside)	(Ca, Mg, Fe, Mn)$OSiO_2$	5–6	3·2–3·5	V	C. Ge. Gr. B. Br.
Quartz (smoky, rose, amesthystine, milky, rock crystal, false topaz, etc.)	SiO_2	7	2·5–2·8	III	C. Y. V. Gr. B. R.
Rammelsbergite	(Ni, Co, Fe)As_2	5·5	7·1–7·2	IV	W.
Realgar	AsS	1·5–2	3·4–3·6	V	R. Y.
Retinite	Fossil resin	1·5	1·01–1·5	O	Br. Y.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Rhodochrosite	$Mn(Ca)O, CO_2$	3·5–4·5	3·4–3·7	III ×	R. W. Y. Br.
Rhodonite	MnO, SiO_2	5·5–6·5	3·4–3·7	VI	R. Br. Ge. Y.
Ripidolite (clinochlore)	$6Mg(Fe)O, Al_2O_3, 3SiO_2 + 4H_2O$	2–2·5	2·65–2·78	V	Ge. R.
Rocksalt (halite)	$NaCl$	2–2·5	2·25	I	Gr. R. Bl. W. Y.
Roselite	$(CoCa)_3(AsO_4)_2, 2H_2O$	3·5	3·58	VI	R.
Ruby	Al_2O_3	9	3·9–4·1	III ×	R.
Rutile	TiO_2	6–6·5	4·18–4·25	II	Br. B. R. Y. Bl. V. Ge.
Salammoniac	NH_4Cl	1·5–2	1·52	I	W. Y. R. Br.
Samarskite	$(U Fe, Cb)_2,O_3, WO_3, SnO_2, ZrO_2, ThO_2, (Fe, Cu, Mg. Ce, Ca, Y)O$	5·5–6	5·6–5·75	IV	B.
Sanidin	$K_2O, Al_2O_3, 6SiO_2$	6–6·5	2·5–2·6	V	Gr. C.
Sapphire (corundum)	Al_2O_3	9	3·9–4·16	III ×	Bl.
Sarcolite	$Al_2O_3, Na_2O, CaO, SiO_2$	6	2·5–2·9	II	R.
Sartorite	PbS, As_2S_3	3	5·39	IV	Gr.
Sassolite	$Bo_2O_3, 3H_2O$	1	1·48	VI	C. Y. Gr.
Satinspar (gypsum)	$CaO,SO_3,2H_2O$	1·5–2	2·33–2·7	V	C. Y. Br. R. Gr. Bl.
Scapolite (wernerite)	$Al_2O_3, CaO, 2SiO_2$	5–6	2·6–2·8	II	C. W. Ge. Gr. B. R.
Scheelite	CaO, WO_3	4·5–5	5·9–6	II ×	C. Y. R. Br. Ge.
Schillerspar (decomposed bronzite)	$3MgO, 2SiO_2, 2H_2O$	3·5–4	2·6–2·8	O	Ge. Br. Y.
Scolecite	$CaO, Al_2O_3, 3SiO_2, 3H_2O$	5–5·5	2·2	V	C.
Scorodite	$Fe_2O_3, As_2O_5, 4H_2O$	3·5–4	3·1–3·3	IV	Ge. Br.
Selenite (gypsum)	$CaO, SO_3,2H_2O$	1·5–2	2·33	V	C.
Senarmonite	Sb_2O_3	1·5–2	5·2–5·3	I	Ge. Br. Y. W.
Sepiolite (meerschaum)	$2MgO, 3SiO_2 + 2Aq$	2–2·5	0·98–1·2	O	W. Gr. Y. R.
Serpentine	$3MgO, 2SiO_2, 2–3H_2O$	2·5–4	2·5–2·6	IV?	Ge. Br. Y.
Siderite (chalybite)	FeO, CO_2	3·5–4·5	3·7–3·9	III ×	C. Gr. R. Br. Ge.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Sillimanite fibrolite)	Al_2O_3, SiO_2 .	6–7	3·23	IV	Gr. Br. Ge.
Silver native	Ag(Au, Cu, As, Sb, Bi, Pt, Hg)	2·5–3	10·1–11·1	I	W. Gr.
Silver glance (argentite)	Ag_2S . . .	2–2·5	7·19–7·36	I	Gr.
Skutterudite	$CoAs_3$ . . .	5·5–6	6·8	I	W. Gr.
Smaltine .	$(Co, Fe, Ni)As_2$	5·5–6	6·4–7·2	Iπ	Gr. W.
Smithsonite .	ZnO, CO_2 . .	5	4–4·5	III ×	Ge. Gr. Br. W.
Soda . .	$Na_2CO_3 + 10H_2O$	1–1·5	1·5	V	W.
Sodalite . .	$3Na_2O, 3Al_2O_3, 6SiO_2, 2NaCl$	5·5–6	2·1–2·4	I	C. Gr. Ge. Y. R. B. W.
Spessartite (garnet)	$(MnFe)_3Al_2Si_3O_{12}$	6·5–7·5	3·7–4·4	I	R.
Sphene (titanite)	CaO, SiO_2, TiO_2	5·5·5	3·4–3·56	V	Gr. Br. Y. Ge. B.
Spessartite (garnet)	$(MnFe)_3Al_2Si_3O_{12}$	6·5–7·5	3·7–4·4	I	R.
Spinel (pleonaste)	MgO, Al_2O_3 .	8	3·5–4·1	I	R. Bl. Ge. Y. Br. B.
Spodumene .	$3(Li_2,Na_2,Ca)O, 4Al_2O_3, 6SiO_2$	6·5–7	3·1	V	Ge. Gr.
Stannite . .	Cu_2S, FeS, SnS_2	4	4·3–4 5	I	Gr. Y.
Staurolite .	$4(Fe, Mg)O, 8Al_2O_3, 7SiO_2$	7–7·5	3·4–3·8	IV	Br. R. B.
Steatite(talc, soapstone)	$3MgO, 4SiO_2, H_2O$	1–1·5	2·6	O	Ge. Gr. W. Br. Y.
Stephanite .	$5Ag_2S + Sb_2S_3$.	2–2·5	6·2–6·3	IV	B.
Stibnite (antimonite)	Sb_2S_3 . . .	2	4·5	IV	Gr. B.
Stilbite . .	CaO, Al_2O_3, $6SiO_2$, $5H_2O$	3·5–4	2·1	V	C. Y. B. R. W.
Stilpnomelane	(Fe, Ca, Mg, K_2)O, Al_2O_3, SiO_2, H_2O	3–3·4	3–3·4	?	B. Y.
Stolzite . .	PbO, WO_3. .	3	7·9–8·1	IIπ	Gr. Y.
Stromeyerite	Ag_2S, Cu_2S. .	2·5–3	6·2–6·3	IV	Gr. B.
Strontianite .	SrO, CO_2 . .	3·5–4	3·6	IV	C. Ge. W. Y. Br.
Succinite (amber)	Fossil resin .	2–2·5	1·0	O	W. Y. Br. R.
Sulphur native	S	1·5–2·5	2·072	IV	Y. R. Ge.
Sylvanite .	$(Au, Ag)Te_2$.	1·5–2	7·9–8·3	V	Gr. W. Y.
Sylvite . .	KCl . . .	2	1·9–2	I	C.
Syngenite (kaluszite)	$K_2O, SO_3 + CaO, SO_3 + H_2O$	2·5	2·6	V	C.
Tachydrite .	$CaCl_2, 2MgCl_2, 12H_2O$	2	1·9–2	III ×	C. Y.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Talc (steatite, soapstone)	3MgO, $4SiO_2$, H^2O	1–1·5	2·5–2·8	IV	Ge. Gr. W.
Tantalite .	FeO, Ta_2O_5 .	6–6·5	7–8	IV	B.
Tennantite .	$4Cu_2S$, As_2S_3 .	3·5–4	4·3–4·5	I	Gr. B.
Tenorite (melaconite)	CuO . . .	3	6·25	IV	Gr. B.
Tetradymite	$2Bi_2Te_3 + Bi_2S_3$	1·5–2	7·2–7·9	III ×	Gr. W.
Thenardite .	Na_2O, SO_3	2·5	2·73	IV	C.
Thomsonite (comptonite)	$(CaNa_2)O, Al_2O_3$ $2SiO_2$, $5H_2O_3$	5–5·5	2·2–2·4	IV	C. Br.
Thorite . .	$ThSiO_4$, $2H_2O$.	4·5–5	5–5·4	II	Y. Br. B.
Tile ore . .	Limonite + Cuprite	—	—	O	R. Br.
Tinkal (borax)	$Na_2B_4O_7 + 10aq$	2–2·5	1·7	V	C. W. Gr. Bl. Ge.
Titanite (sphene)	CaO, SiO_2, TiO_2	5–5·5	3·4–3·56	V	Y. Ge. Br. Gr. B.
Topaz . .	$5(Al_2)SiO_5 +$ Al_2SiF_{10}	8	3·4–3·6	IV	Y. Br. R. Ge. Bl. C.
Torbernite (uranite)	CuO, $2U_2O_3$, P_2O_5, $8H_2O$	2–2·5	3·4–3·6	II	Ge.
Touchstone .	SiO_2 impure with Fe, etc.	7	2·8	O	B.
Tourmaline .	(Al, Fe, Mn, Mg)SiO_3, B_2O_3	7–7·5	2·9–3·3	III ×	B. Br. R. Bl. Ge.
Tremolite (amphibole)	$(CaMg)OSiO_2$.	5–6·5	2·9–3·1	V	W. Gr.
Tridymite .	SiO_2 . . .	7	2·28–2·3	III	C. W.
Triphylite .	(Mn, Fe)$LiPO_4$	5	3·5–3·6	IV	Ge. Gr. Br.
Triplite . .	(Mn, Fe)$_3P_2O_8$, (Mn, Fe)Fl_2	5–5·5	3·4–3·8	IV	Br. B.
Tripolite (diatomaceous earth)	SiO_2 . . .	Soft	1·9–2	O	W.
Trona . .	$(Na_2O)_2(CO_2)_3$, 3–$4H_2O$	2·5–3	2·11	V	C. Gr. Y.
Tungstite .	WO_3 . . .	Soft	3·5–4·5	IV	Y. Ge.
Turgite . .	$H_2Fe_2O_7$ . .	5–6	4·1–4·6	O	R. B. Br.
Turquois .	$2Al_2O_3$, P_2O_5, $5H_2O$	6	2·6–2·8	O	Bl. Ge.
Ulexite . .	Na_2O, 2CaO, $5Bo_2O_3$, $14H_2O$	Soft	1·6–1·8	?	W.
Ulmannite .	NiSbS . . .	5–5·5	6·2–6·5	I ×	Gr. B.
Uraninite (pitchblende)	$U(U_2)O_4$ . .	3–6	4·8–8	I	B.
Uranite (torbernite)	$CuO2U_2O_3P_2O_5$ $8H_2O$	2–2·5	3·4–3·6	II	Ge.
Uranochalcite	U_2O_4(Fe, Cu, Ca)O, SO_3, H_2O	2–2·5	3·19	IV	Ge.
Valentinite .	Sb_2O_3 . . .	2·5–3	5·6	IV	W. Y. Gr. R. Br.

LIST OF MINERALS (*continued*)

Name	Composition	Hardness	Specific Gravity	Crystalline System	Colour
Vanadinite .	$3Pb_3V_2O_8 + PbCl_2$	2·7–3	6·6–7·2	IIIπ	Y. Br. R.
Vauquelinite	$3(PbCu)O, Cr_2O_3$	2·5–3	5·5–5·8	V?	Ge. Br. B.
Vesuvianite (idocrase)	$2(Al_2Ca_3)O_3, 3SiO_2$	6·5	3·3–3·45	II	Ge. Br. Bl. Y.
Vivianite .	$3FeO, 2P_2O_5, 8H_2O$	1·5–2	2·58–2·68	V	Bl. Ge. W.
Volborthite .	$(Cu, Ca)V_2O_5, H_2O$	3–3·5	3·5	?	Ge.
Wad . .	$2MnO_2 + aq$? .	0·5–6	3–4·2	O	Br. B.
Wagnerite .	$2MgPO_4, MgFl_2$	5–5·5	3	V	Y. Gr.
Wavellite .	$2Al_2P_2O_8 + H_2Al_2O_6 + 9aq$	3·25–4	2·33	IV	C. Gr. Y. Ge. Br. B.
Willemite .	Zn_2SiO_4 . .	5·5	3·9–4·3	III	Ge, R. Gr. V. Br. C.
Witherite .	BaO, CO_2 . .	3–3·75	4·3–4·35	IV	C. W. Y. Ge.
Wöhlerite .	$(Ca, Na_2, Fe, Mn)O, SiO_2 Nb_2O_5, ZrO_2$	5·5	3·4	V	Y. Br.
Wolfram .	$(Fe, Mn)O, WO_3$	5–5·5	7·1–7·5	V	Gr. B.
Wollastonite	CaO, SiO_2 . .	4·5–5	2·8	V	C. Y. Gr.
Wulfenite .	PbO, MoO_3 .	3	6·0–7·1	IIπ	C. Gr. Ge. R. Y. Br.
Wurtzite .	ZnS. . . .	3·5–4	3·9–4	III	Br. B.
Xanthophyllite	$2(Mg.Ca)_3SiO_5 + 3Al_2O_3, SiO_2$	4·5–5·5	3	V	Y. Ge.
Xanthosiderite	$Fe_2O_3, 2H_2O$.	2·5	—	?	Y. Br. R.
Xenotime .	$(Y, Ce), PO_4$.	4·5	4·5	II	Br. R. Y. Gr.
Yttrocerite .	$2(9CaF_2 + 2YF_2 + CeF_2) + 3aq.$	4–5	3·45	?	W. Gr. V. R.
Yttrotantalite	$(Y, Ca, Fe)_2, Ta_2O_7$	5–5·5	5·4–5·9	IV	B. Br.
Yttrotitanite	$SiO_2 + TiO_2 + Al_2O_3 + Fe_2O_3 + CaO + YO + CeO$	6–7	3·5–3·7	V	Br. B.
Zaratite . .	$NiO, CO_2, 2Ni(OH)_2, 4H_2O$	3	2·6–2·69	?	Ge.
Zincite . .	ZnO . . .	4–4·5	5·4–5·7	III	R. Y.
Zircon . .	ZrO_2, SiO_2 . .	7·5	4–4·75	II	R. Br. Y. C. Gr.
Zoisite . .	$Ca_4(Al_2)_3H_2Si_6O_{26}$	6–6·5	3·1–3·4	IV	C. Gr. Ge. Y. Br. B.
Zwieselite (triplite)	$(Mn, Fe)PO_4, (Mn, Fe)Fl_2$	5–5·5	3·4–3·8	IV	Br. B.

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (H. Emanuel)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
DIAMOND, white, pink, yellow, red, blue, green, black, orange, brown, opalescent **BOART CARBONATE** (compact massive variety)	Adamantine; reflects prismatic colours None	3·4 to 3·6	Scratches all other precious stones	10	Pure carbon	Cubical	Cube, octahedron, rhombic dodecahedron, tetrahedron, hexa-octahedron	Single	White 2·455, Brown 2·487	0·38	Acquires positive electricity by friction; non-conductor of electricity	Infusible; volatilized by long continued heat	Transparent and translucent Carbonate opaque
SAPPHIRE, white, blue, violet **RUBY**, pink, red, violet-red **TOPAZ, ORIENTAL**, yellow **AMETHYST, ORIENTAL**, purple, violet **EMERALD, ORIENTAL**, green, generally pale	Vitreous very lively	3·9 to 4·2	Scratched by a diamond; scratches all others	9	Alumina 98·5 Oxide of iron 1·0 Lime 0·5	Hexagonal	Hexagonal prism; often pointed at each end	Double in a small degree	1·765	0·026	Acquires electricity by friction and retains it several hours	—	Transparent

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (*continued*)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
CHRYSOBERYL, or **ORIENTAL CHRYSOLITE**, bright, pale-green, greenish-yellow, reddish-brown **ALEXANDERITE**, when exhibiting a reddish transmitted light **CYMOPHANE** or **CHRYSOBERYL CAT'S EYE**, when showing an opalescence like a cat's eye	Vitreous sometimes pearly	3·0 to 3·8	Scratched by sapphire, etc. scratches quartz readily	8·5	Alumina. 80·2 Glucina 19·8 (Trace of peroxide of iron; of oxide of lead and copper, depending on colour and locality)	Rhombic	In flat hexagonal crystals; generally in rolled pebbles	Double	1·760	0·033	Acquires electricity by friction, and retains it several hours	Infusible alone	Transparent and semi-transparent
SPINEL, dark red, white, blue, green **PLEONASTE** or **CEYLANITE**, black **RUBICELLE**, orange **BALAS RUBY**, rose-red	Vitreous	3·8	Scratched by sapphire; scratches quartz readily	8	Alumina 69·01 Magnesia 26·21 Protoxide of iron 0·71 Silica 2·02 Oxide of chromium 1·10	Cubical	Octahedron, rhombic dodecahedron octahedron, tri-octahedron	Single	1·755 to 1·810	0·040	—	Infusible alone	Transparent translucent

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (*continued*)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
TOPAZ, white, greenish, yellow, orange, cinnamon, bluish, pink	Vitreous	3·5 to 3·6	Scratched by sapphire; scratches quartz easily	8	Silica 34·01 Alumina 58·38 Fluorine 15·06 Traces of metallic oxides	Rhombic	Right rhombic prism, octahedral rhombic prism	Double in a slight degree	1·635	0·025	Acquires electricity by friction and heat	Infusible	Transparent translucent
EMERALD, fine green **BERYL** or **AQUAMARINE**, pale sea-green, blue, white, yellow, rarely pink	Vitreous	2·67 to 2·75	Scratched by spinel, scratching quartz (specimens vary)	7·5 to 8	Silica 68·50 Alumina . 15·75 Glucina . 12·50 Oxide of iron 1·00 Lime . 0·25	Hexagonal	Hexagonal prism	Double (very feeble)	1·585	0·026	Acquires positive electricity by friction	Slightly fusible before the blowpipe	Transparent

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (*continued*)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
HYACINTH or **ZIRCON**, brownish-yellow, brownish-red, cinnamon **JARGOON**, various shades of green, yellow, white, brown	Vitreous (almost adamantine)	4·07 to 4·70	Scratches quartz slightly	7·5	Silica . 33·0 Zirconia 66·8 Peroxide of iron 0·10	Tetragonal	Long and short square prisms, Long square octahedron, The prisms often doubly terminated with square pyramids	Double in a very high degree, especially in the Jargoon of Ceylon	1·990	0·044	Acquires positive electricity by friction	Infusible before the blowpipe	Transparent to opaque

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (*continued*)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
GARNET	Vitreous inclining to resinous	3·5 to 4·3	Scratches quartz slightly	6·5 to 7·5	Silica 38·25 Alumina . 19·35 Red oxide of iron 7·33 Lime 31·75 Magnesia 2·40 Protoxide of manganese . 0·50	Cubical	Rhombic dodecahedron, rhombic dodecahedral cube, trapezohedron, hexaoctahedron	Simple	1·759	0·033	Acquires positive electricity by friction	Fusible before the blowpipe	Transparent, opaque
ALMANDINE, violet-red **CARBUNCLE**, red, brownish **CINNAMON STONE**, white, yellow-orange **PYROPE**, vermilion or Bohemain garnet	Rarely asteroid												
TOURMALINE green, red, brown, yellow, blue, black, sometimes white	Vitreous	2·99 to 3·3	Scratches quartz slightly	7 to 7·5	Fluorine . 2·28 Silica 38·85 Boracic acid 8·25 Alumina . 31·32 Red oxide of iron 1·27 Magnesia 13·89 Lime . 1·60 Soda . 1·28 Potash 0·26	Hexagonal	Obtuse rhombohedron, hexagonal prisms	Double	1·625	0·028	Acquires positive and negative electricity by friction and heat	Fusible	From transparent to opaque

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (*continued*)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
QUARTZ or **ROCK CRYSTAL**, white **AMETHYST**, violet **CAIRNGORM**, yellow, brown **CHRYSOPRASE**, fine apple green **CAT'S EYE**, having *chatoyant* reflection **PLASMA**, deep olive green **JASPER**, yellow, red, green, black, brown **BLOODSTONE**, dark green with red spots **CARNELIAN**, red, white, yellow **AGATE**, various colours **ONYX**, having brown, black, and white layers	Vitreous	2·65	Scratches glass	7	Silica 91·37 Alumina trace Amethyst Silica 97·50 Alumina . 0·25 Red oxide of iron 0·50 Oxide of manganese 0·25	Hexagonal	Hexagonal prism and pyramid	Double	1·549	0·026	Acquires positive electricity by friction	Infusible	Transparent and translucent (many varieties nearly opaque)

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (*continued*)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
SARDONYX, having red or brownish and white layers **MOCHA-STONE**, having infiltrated oxides of iron or manganese producing dendritic appearances													
CHRYSOLITE **PERIDOT**, olive green **OLIVINE**	Vitreous	3·3 to 3·44	Scratched by quartz	6 to 7	Silica 39·73 Magnesia 50·13 Protoxide of iron . 9·19 Oxide of nickel 0·32 Oxide of manganese 0·09 Alumina . 0·22	Rhombic	Generally in rolled grains and pebbles	Double	1·660	0·033	Acquires electricity by friction	Infusible	Transparent and translucent

TABLE OF THE DISTINGUISHING CHARACTERISTICS OF GEMS (*continued*)

Name and Colour	Lustre	Specific Gravity	Hardness	No. in Scale of Hardness	Composition	System of Crystallization	Form of Crystal	Refraction	Refractive Index	Dispersive Power	Electric Properties	Fusibility	Diaphaneity
TURQUOISE, blue, green, white	Vitreous	2·62 to 3	Scratches glass feebly	6	Phosphoric acid 27·34 Alumina . 47·45 Oxide of copper . 2·05 Oxide of iron 1·10 Oxide of manganese 0·50 Phosphate of lime 3·41 Water 18·18	None	None	—	—	—	None	Infusible	Opaque translucent at edges
OPAL, colourless, red, white, green, grey, black, yellow, (iridescent)	Vitreous inclining to resinous	2·0 to 2·3	Scratches glass slightly	5·5 to 6·5	Silica 91·32 Water 8·68 Traces of mineral colouring matter	None	None	—	—	—	—	Infusible	Semi-transparent

Valuable Minerals, Their Occurrence, Associated Minerals, and Their Uses

The value of a mineral depends on the use to which it can be put and the quantity available. A deposit which is useless to-day for want of cheap transport, water, or market, may prove to be valuable in the future when conditions change. Some minerals would be more extensively used if they were available in larger quantities or other uses were found for them. A comparatively cheap metal, such as iron, is of more value to mankind than a more expensive metal such as gold. An ore too low in metal to be worth smelting by itself may still be acceptable if it contains fluxes required for another type of ore with which it can be blended, for that is better than using a barren flux. If a mine produces two grades of ore, one high, the other too low for the market, it may pay the miner to add the high grade to the low to bring it up to the marketable grade. The lowest percentage paid for in an ore depends on the purpose for which it is required. For instance, chromite may be wanted for its metal content or as a refractory. Manganese ore may be required to smelt with iron ore, or for the manufacture of chlorine.

The associated minerals mentioned need not necessarily occur with the ore in question, but one or more of them may be associated with them under certain conditions. Some associated minerals are harmful to the one sought, but if they can be separated mechanically beforehand they may both be sold at a profit.

The percentage of metal or salt that a mineral contains (given below) is the theoretical percentage, and is only intended as a rough guide, since minerals are seldom found pure in nature. Prices cannot be stated, as they vary daily according to the supply and demand.

Aluminium. There are only two ores of aluminium at the present day, viz. bauxite, which is the more important, and when pure contains 73·9 per cent alumina, equivalent to 39·3 per cent aluminium; and cryolite which, when pure, contains 13 per cent aluminium. Bauxite is formed from the decomposition of igneous rocks, such as diabase, basalt, volcanic ash, and recent lavas. It occurs in beds, either in its original position or it may have been transported. Cryolite is found in veins associated with blende, iron pyrites, and fluorspar. If bauxite is required for the extraction of aluminium, it should contain at least 50 to 52 per cent of alumina and not more than 3 to 7 per cent silica. When used for making abrasives (alundum and aloxite) it should be low in iron and titanium, and contain not more than 5 per cent silica. For the manufacture of aluminium chemicals, the alumina content must be high, while the iron and titanium must be low. Bauxite itself is used in the manufacture of Ciment Fondu, which is used instead of ordinary Portland cement in cases where it is required that the cement should set quickly. Bauxite is also used as a basic lining for furnaces. Cryolite is used for making a white glass to imitate porcelain. As metal aluminium is so light, it is used in structures where lightness is necessary, e.g. aeroplanes. It is also used in the manufacture of various utensils and instruments, as well as for plating. It forms alloys with copper and nickel, and is used for overhead high-voltage transmission lines.

Alunite. This mineral is usually found in beds. It is used in the manufacture of potash alum. Alum is used for sizing paper, in tanning, and as a mordant in dyeing.

Antimony. Antimony minerals occur in veins, often associated with pyrites, quartz, and gold. The chief antimony minerals are antimonite (stibnite), containing 71·75 per cent antimony; cervanite, containing 80·68 per cent Sb; and valentinite with 83·56 per cent of the metal. It forms important alloys, e.g. type metal, pewter, Britannia metal, anti-friction metal, solder, and for hardening shot. It is also used as a pigment (orange, yellow, and red). The required standard is 50 per cent metal.

Arsenic. Found in veins, especially in those containing lead or copper. The ore is mispickel or arsenical pyrites, 46 per cent arsenic. It is sometimes associated with silver and gold. Paris green is used as a poison for vermin and weeds. Used in the manufacture of opal glass, pigments (green, yellow, and red), and in textile printing.

Asbestos. There are two kinds of asbestos: (*a*) a fibrous variety of the hornblende or pyroxene group; (*b*) a fibrous variety of serpentine, which is the more common. The latter occurs in irregular veins up to 6 in. wide in serpentine rock, but the most workable veins seldom exceed 2½ in. in width. The larger veins are not necessarily the best, as the fibres are generally subdivided at right angles to the length of the fibres. The associated minerals may be chromite, magnetite, or quartz. To be of economic value, the essentials of asbestos are length and fineness of fibre, tensile strength, flexibility, and power to resist fire. It is of less value if coloured. Serpentine asbestos is most suitable for spinning and weaving, while the hornblende asbestos is better for resisting heat and acids. Long fibre asbestos is used for spinning and weaving into yarns, tape, and cloth. It is used for packing piston glands, for salvage blankets, fire screens for theatres, and fire-proof suits. The shorter fibres are used for asbestos sheeting, millboard, and paper. The powder variety is used for gas stove blocks, fire-proof paints, and insulators for boilers and steam pipes.

Asphalt. Asphalt is a semi-fluid solid, the residue of petroleum. It occurs in lakes and springs, but is most common as an impregnation in sandstone and limestone. Used as a filler for pavements, for roofing felt, damp courses for buildings, water- and acid-proof paints, and for insulating purposes.

Barytes. Also known as heavyspar. It occurs in veins associated with galena, zinc blende, iron pyrites, iron and manganese oxides, and fluorspar. The common impurities are quartz, galena, calcite, and iron stains. The standard is 89 per cent of the mineral. Used in the paint industry; for loading soap, paper, rubber, and linoleum; in sugar refining; and to give a green colour to fireworks.

Bentonite. This is a clay that was originally volcanic dust, which settled under water. It is found in beds. It has the property of swelling when wet. The best quality absorbs water up to over ten times its bulk. Used for refining oils and fats, for foundry moulding sand, for preventing leaks in vats, in oil-well drilling, for clarifying turbid water, and purifying sewerage.

Beryllium (*Glucinium*). This is an accessory mineral in pegmatite where it is associated with cassiterite, wolfram, felspar, and mica. Beryl is the only mineral from which beryllium is obtained. The percentage of beryllium oxide varies in the beryl itself, but the theoretical amount is nearly 14 per cent equal to 5 per cent berylium. Its chief use is in the manufacture of strong heat-treated copper-brass alloys used for electrical contacts, clips, small springs, diaphragms, etc.; also in the form of oxide in the fluorescent lamp, X-ray, cathode-ray, and television industries. The transparent variety is used as a gem under the name of emerald and aquamarine.

Bismuth. The chief ores are bismuth-glance, containing 81·4 per cent bismuth; bismutite, 75 per cent bismuth; and bismuth ochre, 89·5 per cent bismuth; native bismuth is also found. It occurs in veins in granite and altered slate, generally associated with ores of nickel, cobalt copper, lead, gold, silver, tin, wolfram, molybdenum, also mispickel and quartz. Used for making certain alloys (fusible metal, soft solder); in medicine; also porcelain and glass painting.

Borax. Occurs in sedimentary beds, mostly in clay and shale. It is largely derived from volcanic ash. It is associated with common salt, soda, glauberite, gypsum, anhydrite, and calcite. Used in laundering; as a flux; to preserve starch and size from mildew; antiseptic; is a constituent in porcelain enamels; as a glaze for paper and linen; in soap; and as a pigment (borate of chromium). Silicon boride, an abrasive, Ferro-boron.

Bromine. This is obtained from brines and deposits of salt. It is used as a chemical reagent, for medicinal purposes, and in photography.

Cadmium. Greenockite contains 77·7 per cent cadmium, but cadmium is chiefly obtained as a by-product from zinc ores which have a coating of greenockite, for it does not occur in sufficient quantities to be worked for itself. Used in the manufacture of certain alloys; in the manufacture of electric lamps of the mercury vapour type; anodes in electric plating; colouring glass, enamels, and pottery; calico printing; and in dentistry.

Cerium. Mostly obtained from alluvial deposits. The minerals from which it is extracted are monazite, cerite, orthite, and pyrochlore. It is used in the manufacture of a special kind of glass for furnace operators, to prevent eye injury; for incandescent gas mantles; to illuminate the path of a fired shell; and photo-electric cells for burglar alarms, etc.

Chromium. The only ore is chromite, which contains 68 per cent chromic oxide. It occurs in irregular masses due to magmatic segregation mainly in periodotite rock. There are few high-grade deposits (over 55 per cent chromic oxide). Chromite itself when between 40 and 50 per cent is mostly used for metallurgical purposes as a refractory: in this case ferric oxide is limited to 15 per cent and silica to 6·8 per cent. The purer ores with a minimum of 47 per cent chromic oxide are used in the chemical and steel industries: the silica permitted being 1 to 1·7 per cent, sulphur 0·5 per cent, and phosphorus 0·2 per cent; it is used for pigments (orange, yellow, red, green, blue, brown, and black); its salts are used in electric batteries; tanning, and plating. The standard for chromium is based on 40 per cent chromic oxide.

Clays. All clays are of secondary origin, and in the moist state are plastic. Their value depends on their physical and chemical properties. The fusibility of clay is increased with the presence of potash, soda, iron, lime, magnesia, and phosphates. When a clay is high in plasticity, it is known as "fat"; when low, it is known as "lean." Clays are given names according to the uses to which they are put, such as brick clay (generally decomposed shale); fire-clay (generally found beneath coal seams) should not contain more than 3 or 4 per cent of combined iron and alkalis; China clay or kaolin, the residue from the decomposition of rocks containing felspar, may occur in veins derived from pegmatite, etc., or as a blanket; pipe-clay; terra-cotta; saggar clay, etc. Clays may be residual in place, or transported. Clays may have to be mixed in order to obtain certain results. Clays are used for making common bricks, firebricks, drain pipes, tiles, earthenware goods, terra-cotta ornaments, crucibles, porcelain, etc.

Coal. Strictly speaking, coal is not a mineral, but is generally classified as one for the sake of convenience, as it is mined on a large scale. Coal is a solid fuel which occurs in seams or beds, formed from the fossilized remains of vegetable matter. There are different kinds of coal. Anthracite contains from 93 to 95 per cent carbon. Bituminous coal contains 75 to 90 per cent. Brown coal or lignite has about 67 per cent carbon and contains much water. Where there is one seam there are generally more in the same basin; these collectively, together with the intervening rocks, are called coal measures. Coal is found in all formations from Paleozoic to Tertiary times, but some periods are more favourable for their formation than others. Bituminous coal is found mostly in the Carboniferous period and brown coal in the Tertiary. Bituminous coal is divided into caking or coking coal, and non-caking. In places, bituminous coal has been seen to merge into anthracite. Brown coal is formed from a higher class of vegetation than that which formed bituminous coal. Coal is a most important substance in the welfare of a country, for on it many industries depend. Its chief use is for generating steam; producing gas; and several by-products, such as aniline dyes, tar, etc.

Cobalt. Asbolite or cobaltiferous wad contains from 2 to 15 per cent metallic cobalt and is the chief source of cobalt. Cobalt glance contains 35·4 per cent cobalt and smaltine has 28·1 per cent cobalt. Asbolite is mostly found in eruptive rocks, frequently in serpentine, where it occurs in irregular bunches associated with clay, but does not go down very far. Cobalt glance and smaltine occur in veins in crystalline rocks. Cobalt is used as a pigment (blue) especially for china; plating; and in the steel trade.

Copper. Copper ores mostly occur in veins which may be in igneous rocks (porphery, granite, or diorite) and in sedimentary rocks. In some places it is found in beds of shale. There are numerous copper ores, most of which are secondary. The primary ore is copper pyrites containing 34·6 per cent copper. Malachite or green carbonate has 57·5 per cent copper; azurite or blue carbonate contains 55·26 per cent copper; covellite or indigo copper has 66·5 per cent; atacamite 59·5 per cent; tetrahedrite 35 to 48 per cent; chalcocite or copper glance 79·8 per cent; cuprite or red oxide 88·8 per cent; melaconite or black oxide 79·85 per cent; and native copper, which sometimes occurs in large masses, which take much time and trouble to cut up. The presence of bismuth and zinc reduces the value of copper. Copper is used for many well-known purposes, the chief of which are sheet, wire, tubes, several alloys (brass, bronze, yellow metal, gun metal, bell metal, etc.); extensively used in the electrical industry; coins; and for blue and green pigments.

Corundum. Corundum occurs in veins in syenite, gneissoid mica schist, hornblende-gneiss, chlorite-schist, serpentine, olivine rocks, and crystalline limestone: it is often found in loose gravel. Three varieties are recognized: (*a*) The transparent crystal, free from flaws, used as gems which are given different names according to its colour; the blue is called sapphire, the red ruby, the green oriental emerald (not to be confused with the mineral called emerald which is a pale green variety of beryl), the purple is known as oriental amethyst (not to be confused with the purple variety of quartz), the yellow variety called oriental topaz (not to be confused with the mineral known as topaz or the yellow variety of quartz). (*b*) Corundum used as an abrasive. (*c*) Emery, a dark granular variety also used as an abrasive.

Diamond. Diamonds appear to have originated in the necks of old volcanoes. Such rocks have to be weathered to obtain the diamonds without damaging them. Diamonds are found associated with enstatite, hornblende, mica, barytes, and magnetite. Diamonds are also found in alluvial associated with garnets, zircon, disthene, chrom-diopside, gold, momazite, topaz, quartz, rutile, chromite, ilmenite, platinum, and iridium. Although such minerals come from the same water-shed, it does not follow that they had their source in the same matrix as the diamond. There are three varieties of diamonds recognized: (*a*) Transparent crystals, free from flaws, used as gems. (*b*) Boart, transparent crystals, but on account of flaws are unsuitable as gems. (*c*) Carbonado or black diamonds, which consist of intermeshed crystals which make them tougher than either of the other two. Boart and carbonado are both used for cutting, grinding, and drilling purposes, but carbonado is better suited on account of its toughness.

Diatomite. Also known as Tripolite. It contains numerous siliceous skeletons of a low form of aquatic plants which live in either fresh or salt water. They form beds in lakes of Pliocene age, often in basalt country. Its value depends mostly on its porosity and its relative chemical inertness. Its porosity is sometimes impaired by deposits of secondary silica in the pores. Employed as an absorbent; for filtering purposes; heat insulation; sound deadening in buildings; metal polishing; manufacture of waterglass; enamels, and glazes.

Felspar. There are several varieties of felspar, but the most important one for commercial purposes, and the most common, is the potash felspar orthoclase, the potash being its most valuable constituent. Found in igneous rocks, chiefly in coarse granite or in veins of pegmatite. It is graded according to the amount of free silica associated with it. The first grade is practically pure. The second grade has between 25 and 30 per cent silica. The third grade has over 30 per cent. Iron is the most objectionable impurity, as it colours any product, thus it must be less than ½ per cent. Felspar is used in the ceramic arts. Soda felspars are used for glazing chinaware and tiles; also as a binder for abrasive wheels.

Fluorspar. Fluorspar occurs as a veinstone in lead and zinc ore deposits. The first class is used in the chemical industry for the manufacture of hydrofluoric acid, and as a catalyst in the manufacture of octane gasoline; enamel for coating iron; in the manufacture of opalescent glass. For these purposes, 98 per cent fluorspar is required; the silica and carbonate of lime must not exceed 1 per cent each. When used in the ceramic industry, 95 per cent fluorspar is required, and the silica must not exceed 2·5 per cent; the maximum amount of ferric oxide allowed is 0·12 per cent. It is employed as a flux in the basic open hearth steel process to eliminate sulphur and phosphorus; for this purpose the minimum amount of fluorspar purchased is 85 per cent.

Fuller's Earth. This is a form of clay. It is mostly used at the present day for clarifying oils, especially mineral oils.

Garnets. Garnets form a group of minerals. They are found in gneiss, mica schist, syenite, hornblende rock, serpentine, and altered limestone; they are also found in alluvials. Some varieties of good colour, when free from flaws, are used as gems, such as pyrope and almandine; but the chief use to which garnets are put is as an abrasive, more especially for wood and leather.

Gems. These may be precious or semi-precious stones. Precious stones are noted for their hardness, lustre, transparency, in most cases for their colour and size within limits, also their rarity. The colour of a gem may change its name. The sapphire (blue), ruby (red), oriental topaz (yellow), all have the same chemical composition. Rock crystal (colourless), amethyst (violet), cairngorm (brown or yellow), are all varieties of quartz. The colour is not always characteristic, for ruby, spinel, and certain garnets are all red, though of different shades. Most precious stones are found in alluvial deposits where they have been freed from their matrix. They are found associated with other minerals which have survived on account of their hardness and toughness, which minerals need not have come from the same source though occurring in the same water-shed. The harder a gem, the brighter its lustre. Among precious stones may be mentioned the diamond, sapphire, ruby, aquamarine, chrysoberyl (oriental cat's eye), alexandrite. Among the semi-precious stones are rock crystal, amethyst, cairngorm, chrysoprase, onyx, agate, bloodstone, spinel, zircon, topaz, peridot, garnet, tourmaline, moonstone, opal, and turquois.

Gold. Gold occurs in veins associated with quartz, calcite, barytes, mica, fluorspar, kaolin, gypsum, iron pyrites, pyrrhotite, mispickel, hæmatite, limonite, copper pyrites, zinc-blende, galena, antimonite, fahlore, cinnabar, pitchblende, etc. Gold also occurs in alluvial deposits associated with platinum, osmiridium, cassiterite, tourmaline, zircon, titaniferous iron, and diamonds. Native gold is the main source of supply. It is sometimes found in nuggets that appear to have been large masses of gold originally in a vein which have been pounded by Nature, thus eliminating most of the brittle minerals associated with it. Gold is soluble in alkaline sulphides found in mine waters, and has been found redeposited on mine timber which has been under water for a considerable time; but that is not likely to account for the formation of nuggets.

Graphite. Also known as plumbago. It occurs in veins and beds in limestone, pyroxenite, gneiss, mica schist, and quartzite. It may be associated with quartz, calcite, felspar, chlorite, or hæmatite. The crystalline or flake variety is found in the older rocks; the amorphous mostly in the younger rocks. Flake graphite is used in the manufacture of refractory articles, such as crucibles, retorts, dippers, stirrers, and nozzles. The standard is 85·9 per cent carbon. It is mixed with a special clay to bind it. Any clay naturally mixed with graphite, and which is not suitable, must be separated beforehand. Amorphous graphite is used for lead pencils, lubricating heavy machinery, foundry facings, polishing iron, paint, and for electrical purposes.

Gypsum. Found in sedimentary beds, due to the evaporation of saline waters. When calcined it is known as plaster of Paris, and is used for wall plasters.

Helium. Obtained from certain rare natural gas fields. Used as a lifting medium for balloons and air-ships.

Iodine. Obtained from nitrate deposits, salt wells, also from the ashes of kelp. Mostly used in the chemical industry.

Iridium. Obtained from alluvial deposits, where it is associated with gold and other minerals generally found in such deposits. Used for the points of gold nibs, for tips of rubber-cutting tools, and for knife edges of delicate balances. It is alloyed with gold, silver, and platinum.

Iron. Iron ores occur both in lodes and beds, as replacement deposits in limestone, and in masses. Bog, lake or spring iron ore can be seen in process of formation at the present day in or near the borders of swamps or chalybite springs. Clay ironstone is to be found in the coal measures as aggregations. Blackband is a clay ironstone containing a considerable amount of carbonaceous matter. When iron ores occur in metamorphic rocks, they are generally hæmatite or magnetite. The chief ores of iron are hæmatite containing 70 per cent iron, magnetite with 72·4 per cent iron, limonite with 59·9 per cent iron, and chalybite (siderite or spathic iron ore) with 48·3 per cent iron. Phosphorus and sulphur are the two elements objected to by iron smelters. They sometimes occur in shoots and may be eliminated in the mining, but their limitations can only be determined by systematic sampling and assaying. Commercially, iron ores are classified as red, brown, magnetic, and carbonate. Iron is the most useful metal we have, though it is being replaced for many purposes by aluminium. Its chief use is for making steel of various sorts employed in construction (rails, girders, plates, sheets, pipes, wire, etc.). Natural ochres, formerly used as pigments (red and yellow) have now given way to those formed artificially as by-products, since the latter are more uniform in tint, it being difficult to supply duplicate orders for the same tint with natural ochres. The price of iron ore is based on 55 per cent of the metal present.

Jade. Occurs in cavities in gneiss, granite, and syenite

Much of the mineral that comes from China called jade is really prehnite. It is used for making buttons, beads, amulets, and is carved into images, cups, and other ornaments.

Lead. Lead ores mostly occur in lodes, irregular deposits, and in beds. It is found in crystalline schists, shale, sandstone, and limestone. It is associated with zinc, silver and copper ores, iron pyrites, baryta, quartz, fluorspar, and calcite. Its chief ores are galena with 86·55 per cent lead; cerussite with 77·53 per cent; and anglesite with 68·32 per cent lead, the two latter occurring in the zone of secondary enrichment. It is made into sheets, pipes, and shot; it is an ingredient of certain alloys (solder, pewter, and type metal); as a pigment (white and red lead); for chemicals (litharge, lead arsenate, etc.).

Limestone. If limestone contains 15 to 18 per cent magnesia it is called magnesium limestone or dolomite. It occurs among sedimentary rocks of all ages, but has not always been formed in the same way. It has a diversity of uses: as a building stone; in the manufacture of Portland cement; mortar; as a flux for smelting furnaces; for road-making; agricultural purposes; manufacture of various calcium salts; water treatment; whitewashing; in the paper industry; tanning; calcium chloride is used as an anti-freeze and for laying dust on highways. Chalk is a variety of limestone, so is lithographic stone.

Lithium. Obtained from spodumene, which contains from 4·5 to 7 per cent lithium. Occurs in pegmatite associated with beryl, columbite, wolfram, cassiterite, tourmaline, and uranium minerals. The metal itself is not used, but its compounds are: bromide and chloride are used in photography, carbonate in medicine, and hydroxide in storage batteries.

Magnesite. Formed from the decomposition of dolomite or from serpentine. Occurs in joints and veins, but they do not go down to any great depth. It is also found in nodular masses in alluvial deposits. Any magnesite containing 90 per cent of carbonate magnesia or over is considered high grade. Impurities in lower grades are silica, alumina, lime, iron oxides, and earthy matter. When required for the manufacture of sorrel cement, or as a filling for paper, the silica may be as high as 12 per cent and there is no limit for lime, but the iron must be below 1 per cent. It takes from 2 to 4 tons of crude magnesite to make 1 ton of calcined magnesium oxide. The mineral is used in the manufacture of imitation marble and for stucco. It is also used as a refractory material for lining converters and open hearth steel furnaces: for these purposes it must not have less than 8 per cent magnesia, but less than 6 per cent silica; there is no limit to iron oxide, of which there is generally but a small amount. The loss on ignition must not exceed 5 per cent. The standard for magnesite is 90 per cent. Magnesite is used as an adulterant for paints, weighting cotton goods, as a bleaching agent, for toilet purposes, in the manufacture of carbonic acid gas; also for Epsom salts. The metal is used in flashlight photography, for pyrotechnic purposes; and the war has brought about its use in incendiary bombs.

Manganese. The ores occur in igneous, metamorphic, and sedimentary rocks, where they occupy superficial veins, joints, and occasional pockets: they are also found in clay deposits. The deposits are generally irregular and seldom go deeper than 300 ft. Below water-level the oxides give place to carbonate or silicate of manganese. Manganese is often closely associated with iron, forming ferruginous manganese ore. The ore is classed as manganese ore if it averages 45 per cent manganese, and as ferruginous ore if between 35 and 45 per cent manganese. The ores of manganese are pyrosulite, with 63·6 per cent manganese, psilomelane 52 per cent, braunite 69·6 per cent, manganite 62·5 per cent, and rhodochrosite 47·4 per cent. The standard for manganese ore is generally taken as 60 per cent manganese oxide, the price going up and down on a sliding scale. The chief use for manganese is for making manganese steel; fine soft ore is not suitable for this work, it should be hard and of moderate size. Chemical ore should be high in oxygen and may contain more silica than metallurgical ore, but carbonate of lime should be limited to 1 per cent. It is used for bleaching purposes, depending on the amount of chlorine it will liberate when treated with hydrochloric acid, some oxides requiring more acid than others. Used for pigments (violet, brown, black); as a depolarizer in dry cells; potassium permanganate is a strong oxidizing agent; manganese dioxide and certain salts are used as dryers in varnish, japan, and printing ink.

Marble. Strictly speaking, marble is an altered limestone, but in the trade the term is also applied to other ornamental building stones. Marbles are given various special names according to their colour, such as verde antique, a cloudy green; the locality in which it is found, such as Carrara, a pure white semi-translucent marble; or its main component, such as encrinal marble, composed largely of the fossil remains of the sea lily. Marbles may occur in any age up to the Trias. There are seldom more than three different colours in a marble, but they may have various shades. In multicoloured marbles, very dark veins and patches which do not shade off are generally weak places and are liable to crumble. If a stone is wet, it gives a good idea what it will look like when polished. Marble is used for interior and exterior decorative work in buildings, tombstones, statuary, butchers' and fishmongers' slabs. Marble in the block is sawed into slabs more than 2 in. thick and is sold by the cubic foot. Slabs 2 in. or less are sold by the square foot.

Meerschaum. This is an alteration product of magnesite or serpentine. It occurs in nodules and is sometimes picked up on the seashore. Used for smokers' articles.

Mercury. Quicksilver, though found in rocks as old as the Silurian, would appear to have been deposited in more recent times. The following minerals are sometimes found associated with cinnabar, barytes, calcite, fluorspar, quartz, gypsum, iron pyrites, antimonite, mispickel and realgar. Quicksilver deposits occur mostly as stockworks, stocks, and veins. Cinnabar containing from 64·93 to 73·53 per cent mercury is practically the only ore of mercury, though a small amount of the metal is found in the native state. It is used in the manufacture of explosives (fulminate), in the metallurgy of gold, in the construction of scientific instruments (barometers, thermometers), silvering mirrors and reflectors, as a pigment (vermilion); also as chemicals (corrosive sublimate, and calomel).

Mica. There are several micas, but the chief one of commerce is muscovite, found in veins of pegmatite, where it appears to have been the first mineral to crystallize out, followed by felspar and, finally, by quartz. Near the surface, mica is weak, cracked, easily deformed, and its transparency spoilt; but its size and quantity give an indication of what may be expected deeper down. The best mica is found in hard rock where it has not been subjected to much water percolation. Crystals, or books of mica as they are called, are not found of commercial value in veins of less than 2 ft wide; on the other hand, some of the largest pegmatite veins do not contain mica of commercial size. The books of mica do not occur in any particular order. The smallest marketable size in blocks is 3 in. by 2 in., but it only pays to mine these when larger sizes are also available. The usual marketable sizes are from 3 in. by 2 in. to 8 in. by 10 in. Any small flakes required are obtained from the trimmings of sheets. The percentage of mica is generally less than one of the rock blasted down, and of the mica extracted, less than 10 per cent if it is of value as sheet mica. Mica is trimmed by the miner, but not cut into shapes and sizes, as he does not know the requirements of the buyer. Mica is sold by the pound. To be valuable, it must be flat, smooth, uniform, free from flaws, able to split well without tearing, and must not be cross-grained. The colour does not matter for some purposes, but white, free from iron stains, is preferred. It is used for electrical insulation purposes, stove fronts, lamp chimneys, scrap is ground and used for lubricating heavy machinery; also for decorative purposes.

Molybdenum. Occurs in quartz veins in granite. Molybdenite is the only ore of molybdenum which, when pure, contains 60 per cent of that metal. Injurious elements liable to occur with molybdenite are arsenic, bismuth, tin, wolfram, antimony, calcium, barium, and phosphorus. More than 2 to 3 per cent copper renders the ore unsaleable; tin and bismuth may be present up to ½ per cent without being penalized. The ore has to be dressed up to 90 per cent to be marketable. Used for making steel for high-speed tools, heavy artillery and rifle barrels, stainless cutlery; alloyed with tungsten to make points for spark plugs; when alloyed with cobalt and chromium, it is hard and has acid-resisting properties; used as a blue pigment for pottery; for several chemicals.

Monazite. This mineral is a phosphate of certain rare metals, namely, cerium, thorium, lanthanum, and didymium. Its value depends on the content of cerium and thorium. The thoria content of monazite from different localities varies considerably from about 1 to 8 per cent. Monazite was originally formed in pegmatite veins, but is mined in river beds and beaches, where Nature has done the preliminary mining and concentration. In alluvial deposits it is found associated with magnetite, ilmenite, chromite, wolfram, scheelite, cassiterite, corundum, gold, garnet, zircon, rutile, topaz, and quartz. The most important use of monazite is for its cerium, which is extensively used for the cores of searchlights and cinema carbons. The thoria is used for incandescent mantles. The presence of iron in the thoria nitrate is objectionable as impairing the luminosity of the mantle as well as its durability.

Nickel. Nickel is closely associated with igneous rocks (granite, diorite, peridotite, and its alteration product serpentine), also in gneiss and schists. Its ores are noumeite, with 25 per cent nickel, niccolite (copper nickel) 43·6 per cent, zaratite 46·8 per cent, pentlandite, a nickeliferous pyrrhotite, up to 20 per cent nickel. Associated minerals are ores of cobalt, copper, lead, silver, and bismuth. Used for making nickel-steel, for certain alloys (monel-metal and German silver), and for plating.

Nitre or Saltpetre. Occurs in beds along the shore line of lakes in dry districts. The calche or raw material is generally sand or gravel cemented together with nitre. The chief impurities besides gravel are common salt, gypsum, and sulphates of potassium, sodium, and magnesium. It is used as a fertilizer; for dry salting; in the manufacture of explosives and fireworks; and various chemicals.

Oil Shale. Also known as kerosene shale, cannel coal, and torbanite. Found on the margin of coal basins. Supposed to have been formed by the accumulation of minute plant bodies, such as fresh water algæ, which lived in the lakes and swamps of ancient times. Its value depends on the number of gallons of oil that can be obtained from one ton of shale. The crude oil on fractional distillation yields various products such as gasoline, benzine, lubricating oil, and paraffin.

Opal. There are several varieties of opal, but the precious or noble opal with its sub-varieties is the only opal of commercial value, and it makes a very fine gem. Opal mining is precarious, for as a rule there is no surface indication of its presence except the class of country in which it is likely to be found. Opals are found in the interior of siliceous ironstone nodules, which are found in kaolin beds of cretaceous formations, also in sandstone and andesite. The points to be considered are the weight and shape of the specimen; relative freedom from flaws; display of colours and fire. Patterns are important: these may be pinfire when the grain is very small, harlequin when the colours are in small squares, or flash fire when the colour shows in a single flash in a very large pattern. The opal should be neither too transparent nor too opaque. Some opals are harder and tougher than others, which is an advantage, as they retain the polish better.

Petroleum. The general opinion is that petroleum had its origin in organic matter. Known oil fields are not confined to any particular age. They are found from the Silurian to the Tertiary, but more especially in the latter. If oil reservoir rocks are present, a dome-shaped structure is favourable for the accumulation of oil. At the highest point, gas is likely to be encountered as being lighter than oil; then comes the oil and, finally, water. Salt or saline solutions nearly always accompany petroleum, probably from marine or brackish shales. A gentle anticlinal structure does not necessarily show on the surface for older formations may be covered by those of a younger age. Among other products obtained from petroleum are gasoline, naphtha, benzine, kerosene, petrol, lubricating oil, petroleum jelly, and paraffin wax.

Phosphates. Apatite is found in veins of the Laurentian age, where it occurs in all kinds of igneous rocks, more especially olivine-gabbro and schists, where it occurs with pyroxine, felspar, scapolite, topaz, copper pyrites, and pyrrhotite. Coprolites (the petrified dung of extinct saurians and fish) are obtained from certain beds and from gravels in some rivers; but the principal supply of phosphate is from phosphate rock formed by the displacement of carbonic acid gas of coral rock by phosphoric acid leached out from the droppings of sea birds. The fresher droppings of birds and bats are known as guano. The basis on which phosphates are sold is tri-calcic phosphate. Phosphate rock, when pure, contains 66·77 per cent tri-calcic phosphate; for the market should not contain less than 40 per cent. It is mostly used in the manufacture of superphosphates for agricultural purposes; iron and alumina are harmful for this purpose and should not be more than 3 per cent of the combined oxides. More than 5 per cent of lime is also objected to as consuming too much acid in the manufacture of superphosphate. Phosphorus is used in the manufacture of phosphor-bronze; matches; and as a poison for noxious animals.

Platinum. This metal occurs in small quantities in olivine-bearing rocks, such as peridotite, dunite, and their alteration products. It is also found in augite-syenite and hornblende-diorite. Minerals sometimes associated with it in veins are iron and manganese oxides, chromite, and niccolite. Commercial platinum was formerly obtained mostly from river, alluvial and sea-beach deposits, but the largest producing source for some years has been the Ontario copper-nickel matte. In Nature it is generally alloyed with osmiridium, palladium, or rhodium. It is used in the construction of various chemical apparatus (evaporation dishes, crucibles, funnel points, spatula, foil, wire); electrical apparatus; as a catalyzing agent in the manufacture of sulphuric acid; jewellery; pins for artificial teeth; and porcelain painting.

Potassium. Occurs in beds of Permian and Tertiary ages, together with rock salt. The principal ores are sylvite, carnallite, and kainite. Used as a fertilizer, in the manufacture of several potassium salts (cyanide, carbonate, sulphate, caustic, chromate, bi-chromate, chlorate, bromide);

used in medicine; manufacture of glass; explosives; electroplating; fireworks; matches; dyeing; and tanning. Sylvite, when pure, has the equivalent of 63·3 per cent of potash, carnallite 16·9 per cent, and kainite 28 per cent.

Quartz. There are several varieties of quartz with different names, according to their colour and patterns, mostly obtained from alluvial deposits, and used as ornaments and for scientific purposes. Crystalline varieties are known as rock crystal (clear transparent), amethyst (purple or violet), rose quartz (pale pink), cairngorm (yellow or brown). Cryptocrystalline varieties are carnelian (flesh-coloured), prase (leek green), plasma (leek green speckled with white), blood-stone or heliotrope (green speckled with red), agate having bands of different colours, onyx, sardonyx, jasper, etc. When broken up it forms sand, which is used for many purposes.

Radium. Radium is found in small quantities in several minerals and is always associated with uranium. The chief sources for commercial purposes are pitchblende and carnotite. Pitchblende is found in granite, and slate altered by the intrusion of granite; also in veins of pegmatite. Carnotite is found impregnating certain sandstones.

Rock Salt. Rock salt occurs in beds, sometimes of great thickness, together with chlorides of potassium and magnesium, and associated with gypsum and anhydrite. It is found from Silurian times to the present day, but more especially in the Tertiary and Mesozoic periods. Salt occurs in depressions of the ground from which there is no escape for the water, except by evaporation. Salt is also obtained from the sea, salt lakes, and brine springs. It is an essential food used in cooking; licks for cattle; dry salting; dairying; metallurgy (chloridizing, roasting); prevention of dry rot; in the manufacture of chlorine, soda, and various chemicals; in the tobacco industry; as a weed killer.

Selenium. This metal is not mined for itself, but is obtained as a by-product from the manufacture of sulphuric acid, in the anode slimes from copper refining, and also in the treatment of gold ores. There is no great demand for this metal. It is used in photo-electric cells, for improving the strength of rubber, for removing the green tint from glass; also in the manufacture of orange-coloured glass and red enamel ware.

Slate. Marketable slates occur mostly in the older formations (Cambrian and Silurian, and even as high up as the Carboniferous), which are highly tilted. The necessary properties are cleavage, which must be perfect, straight, and capable of being split into thin sheets; colour which must be uniform; inability to absorb water, for in cold climates any water in the slate will freeze and, in so doing, expand and chip off flakes, besides porosity gives lichen and moss a chance to grow on slate; for billiard tables; stair treads; window sills; hearthstones; lining for sinks and cisterns; roofing; school slates; paving stones; switchboards, when its electrical resistance must be good. When pulverized, it is used as a filler for paints; in the making of abrasive soap.

Strontium. The ores of strontium are celestine, which contains about 50 per cent strontia and should not contain less than 90 per cent of the sulphate to be of commercial value. It is found in irregular rounded or sheet masses in clay, marl, limestone, dolomite, sandstone, and serpentine interbedded with tuffs. Sometimes associated with gypsum, barytes, and rock salt. Strontianite, containing 70 per cent strontia, occurs in small veins in marl and limestone, but does not live down. Probably derived from solfataras. Its use is limited to its salts, which are employed in desaccharizing beet sugar, molasses; for fireworks and flares.

Sulphur. Sulphur occurs in the native state, also as sulphides (chiefly as iron pyrites). Native sulphur is found in beds varying from a few inches to several feet in thickness in the neighbourhood of both active and dead volcanoes. It also occurs in porous sandstone. Iron pyrites contains 40 per cent sulphur and is generally found in lodes. Used in the manufacture of sulphuric acid; in petroleum refining; pickling iron and steel prior to galvanizing; in the manufacture of soda ash; certain explosives; in kraft paper plants; synthetic rubber; and sprays for vegetable pests (carbon bisulphide, sulphur-lime).

Talc. The foliated variety is known as talc; the massive variety is known as soapstone or steatite, It is an alteration of hornblende, augite, olivine, and diallage, and occurs as veins or beds in crystalline rocks associated with tremolite. Found only in much altered ultra-basic intrusives and in certain metamorphized dolomitic limestone. Used for loading paper; fire-proof paint; talc powder for toilet purposes; spreading on dance floors to make them slippery; acid-proof floors, tanks, and vats; ceramics; switchboards; electric insulators; pipe and boiler insulation; blackboard chalk (french chalk); fire-stones; foundry facings; sizing cotton cloth; in the rubber industry; and for carving figures.

Tantalum. Occurs both in granite and alluvial deposits associated with cassiterite. The ores are tantalite, containing 51·13 per cent tantalic oxide and columbite. It is used for filament wire in lamps and other electrical apparatus, parts of scientific instruments, surgical tools, and nib points.

Tin. Found in veins, pipes, and stockworks which occur in granite, greisen, porphyry, gneiss, and slate. Associated minerals are iron pyrites, wolfram, scheelite, mispickel, bismuth, quartz, mica, tourmaline, felspar, topaz, fluorspar, and beryl. It is also found in deep leads of the Tertiary period and in alluvial deposits of recent date. Nearly all the tin produced is obtained from cassiterite, which, when pure, contains 78·67 per cent tin. The only other tin-bearing mineral is stannite (tin pyrites or bell-metal ore), which carries 27 per cent tin and 30 per cent copper, and is generally treated for its copper contents. Used for coating thin sheets of iron commonly known as tin-plates, which are manufactured into various articles: alloys (solders, fusible metals, bearing metal, type metal; bell metal, pewter, Britannia metal, bronze, gun metal), tinfoil for wrapping up certain foods and tobacco; amalgamated with mercury in the construction of mirrors; weighting silk before dyeing; calico printing; and in the manufacture of opaque glass.

Titanium. This is found in quartz veins which occur in igneous rocks, such as granite and gabbro; in metamorphic rocks, such as gneiss and mica schist; and in sedimentary rocks, such as limestone; also as black sand in streams. The ores of titanium are rutile with 60 per cent titanium, and ilmenite, also known as titaniferous iron carrying various amounts of titanium from 31·6 per cent. The metal has but small use. It can be employed in making high-speed steel; for enamel paints; titanium tetrachloride is used for making smoke clouds and for sky writing; electrodes for arc-lighting; alloys (manganese-tatanium and cupro-tatanium); as a mordant in dyeing; particularly for leather; in glass and ceramic arts; in the manufacture of dentures; and as a yellow pigment.

Tungsten. The ores are wolfram with about 76·57 per cent tungstic trioxide, and scheelite with about 80·5 per cent, which are found in granitic rocks associated with cassiterite, molybdenite, iron pyrites, pyrrhotite, mispickel, copper pyrites, galena, zincblende, bismuth ores, topaz, fluorspar, and mica. Tungsten is used to give greater hardness to high-speed steel; in the production of bronze powder, to make fabrics non-inflammable; and a purple pigment for dyeing.

Uranium. The ore is pitchblende, containing 56·5 to 61·5 per cent uranium oxide. It occurs in veins with ores of lead, silver, or tin. It is a source of radium. Used in atomic bombs;

as a yellow colour for porcelain painting; and for colouring

Vanadium. Found in veins and impregnating sandstone. The ore is vanadinite. Vanadium is found in magnetite, especially the titanic variety, also in certain carbonaceous material such as asphalt and certain coals, and in some rare minerals such as patronite and carnotite.

Zinc. Zinc mostly occurs in veins in granite, gneiss, and shale, and in irregular deposits in limestone. Its associated minerals are lead, silver, and copper ores, iron pyrites, mispickel, magnetic pyrrhotite, specular iron, garnet, epidote, talc, amphibole, calcite, chlorite, and tourmaline. The principal ores of zinc are zincblende (black jack) containing 67 per cent zinc, smithsonite with 52 per cent, willemite 55·6 per cent, and calamine with 54·17 per cent zinc. Metallic zinc is known on the market as spelter. It is largely used for galvanizing sheet iron and wire, in the metallurgy of gold, for electric batteries, castings, alloys (brass, Muntz metal, nickel-silver, anti-friction metal), pigments (white and yellow), and as a mordant.

Zirconium. Zircon is the only ore and it contains 67 per cent zirconia. It is found in crystalline rocks (pegmatite, syenite), metamorphic rocks (gneiss), and sedimentary rocks (limestone), also in alluvial deposits. It is associated with ilmenite, magnetite, monazite, rutile, quartz, and fluorspar. It is used in precious metal refining, and in electric furnaces where exceptionally high temperatures are required; it is relatively inert to molten silica slags and other corrosive agents, but does not withstand the action of basic slags very well; as a clouding agent in enamels; in a small way in the manufacture of incandescent gas or lamp mantles.

MINING

Prospecting

Prospecting is searching for valuable mineral deposits. When one has been found, then it has to be explored to determine its limits and whether it is worth while continuing to work it. If the results are satisfactory, the mineral is mined. These three stages merge one into the other and are often carried out simultaneously in different parts of a mining property.

A prospector may work by himself, with a mate, or in a party under supervision of a geologist. The equipment depends on local conditions and what is sought; anyhow, it is made as light as possible, for unknown mineral deposits are generally in rough unexplored country. The prospector should know something about the mineral sought, the secondary minerals likely to be found near the surface, the rocks in which it is most likely to occur, and the nature of the deposit. The landscape is affected by rocks. Igneous rocks such as granite are generally rugged. Lava flows form long level table-lands, for though originally the lava occupied lowland, when it solidified it is generally harder than the surrounding country and so resists weathering action better than sedimentary rocks, and in time becomes the highland. If a flow of basalt took the place of a stream in which gold or tin had collected, one would have to attack the alluvial from under the basalt by shaft or adit. Igneous dykes, if hard, stand out like walls; if soft, they wear out into chasms. Limestone weathers into jagged pinnacles and often contains cavities, but spaces between the pinnacles may be filled up with soil. Sedimentary rocks, such as shales and sandstones, are generally smooth and rounded.

Any irregularities on the surface of the ground should be noted, as they may indicate the presence of a fault. Salt is deposited in depressions, from which there is no escape for the water except by evaporation. In dry countries, salt may appear from a distance to be water. In other cases, when a bed of salt occurs below the surface, a depression may indicate where some of the salt has been dissolved.

Modern inventions have led to new methods of prospecting. Aerial photography with improved cameras enable us to get a bird's-eye view of the landscape better than can be obtained from the ground, and by this means certain geological features can be studied, such as faults, a change in the rocks, variation in vegetation, colour of the soil, etc.; moreover, one can determine the best way to travel and transport material in undeveloped country. All this saves much time and trouble in determining the most likely places to search for the desired mineral. Aerial prospecting is not intended as a substitute for ground prospecting, but as an accessory.

When seeking tin or gold in hilly country, these being heavy would naturally gravitate from their source to creeks. The prospects are followed up the watercourse until they cease, after which the prospector takes samples systematically of the soil on the sides of the hills and washes the grass roots which may have collected some of the mineral sought. When these prospects cease and no shoad or shed stones are found, trenches are cut at right angles to the supposed strike of the lode until the prospector finds the source of his prospects. Searching for other deposits, such as copper or lead, he at first has to depend largely on vein stone shed from the outcrop and stains of secondary minerals a little below the surface.

Copper shows blue, green, or red stains; lead white, yellow, green, and sometimes red; iron brown or reddish; nickel, green; tin ore already being oxidized does not show secondary mineral stains. All gossans or iron cappings are not indications of sulphides at a depth; when massive they may be the oxidized outcrop of basic igneous rocks in the form of dykes. The gossan of copper or lead lodes are softer, and generally show small cavities which may give an idea of the quantity of valuable minerals which have been dissolved. Minerals generally associated with certain ores may also occur without the latter, so their presence is no guarantee that the ore looked for is there. Fragments of rock from various places in the same water-shed may be found in creeks, not necessarily because they were associated with the mineral sought, but because they were hard, tough, heavy, and stable enough to collect together. In alluvial, one may find gold, platinum, monazite, various precious stones (diamonds, sapphires, rubies, zircon, garnets, etc.), tinstone, chromite, magnetite, besides shot, bullets, buttons, coins, and broken up water-worn blue castor oil bottles, sometimes mistaken for sapphires. When rocks are warmed by the decomposition of sulphides and snow falls on them, the snow will thaw more quickly in such places than away from them.

Sight may in some cases be assisted by smell, as in the case of sulphuretted hydrogen given off at the outcrop of sulphides under certain conditions. Springs containing brine, iron, or carbonic acid gas are worth careful examination. Mineral springs generally kill the vegetation with which they come in contact, whereas fresh water promotes vegetable growth.

Old deserted mines are sometimes worth examining, and much information may be gained by overhauling the mullock heaps with regard to the extent of the workings and the rocks met with underground. Where mining has taken place, all available records should be looked up. Memories of old people who knew the mine when it was being worked are apt to err on exaggeration lines. A grade of ore formerly looked on as unpayable mineralized rock may pay to work by modern methods and improved treatment, especially if transport and water supply conditions are favourable.

Prospecting for oil is too expensive for the ordinary prospector, as it generally entails deep drilling, but a prospector may observe oil floating on the surface of water, though this does not necessarily indicate the presence of oil in payable quantities at a depth; besides, the oil may have come from some other source. An oil prospector requires a fair amount of geological knowledge and may have to make an extensive geological survey. In addition to the oil, it is necessary to have a reservoir to hold it: this is generally a porous sandstone or unconsolidated sand. The reservoir must have a covering which will not allow the oil to escape till tapped: this covering is generally shale free from faults.

In places where granite has come in contact with limestone, the latter is sometimes converted into garnet rock. If there is copper in the mineral water, it is deposited in the numerous joints of the garnet rock, which is more or less green. When the rock is struck it naturally breaks at the joints, and these being coated with green carbonate of copper make the rock look much richer than it really is.

Rock on the surface is naturally more broken up than deeper down. Minerals below water-level are more stable under their conditions than those nearer the surface, where they are subject to oxidation. These secondary minerals derived from primary ores find their way into the weathered surface rock adjoining lodes, thus making the outcrop appear much wider than the true width deeper down. This is known as a 'spew over,' and often causes the prospector much disappointment when he sinks and finds the lode narrower and poorer.

Mining of the future is more likely to be of large low-grade deposits than smaller rich deposits.

Some supposed tests of minerals are worse than useless. Those who confuse hardness with brittleness suggest that a stone, supposed to be a diamond, be placed on an anvil and struck with a hammer. If the specimen is not broken, it is a diamond; but if it breaks, it is not, so no harm is done. No mineral will stand such treatment. Another supposed test for a diamond is that if the specimen will scratch glass it is a diamond; but quartz and many other minerals will scratch glass, so that is no test. One should try the supposed diamond for hardness on a sapphire.

Of recent years, various geophysical methods of prospecting have been introduced. No one method is capable of general application. Very sensitive instruments are required, and great care and skill in their use. Each method has its own limitations regarding the class of work it can undertake, and their value depends on the proper interpretation of the data, which is no easy matter.

The Magnetic Method. The dipping needle and an ordinary compass have long been used for determining the position of strongly magnetic ores, such as magnetite and pyrrhotite. Now a more delicate instrument called a magnetometer is used.

The Gravimetric Method. The force of gravity is not constant. On the surface, above relatively heavy materials, the pull is increased. These measurements are carried out with the Estovo's torsion balance. The process is slow, delicate, and expensive. It may be useful for determining the lines of buried valleys and other structural discordances of the Earth's crust, such as locating faults. So far, its economic use has been mostly confined to determining salt domes and anticlinal structure of oil fields. Its usefulness for depths greater than 1500 ft is questionable.

Electric Methods. There are several electric methods: Spontaneous polarization (Schlumberger's), equipotential line method, D.C. and A.C. electro-magnetic method, and the Radio-wave method. They all depend on the relative electrical conductivity of certain ores and the rocks which enclose them. Ores with a metallic lustre, such as sulphides of iron, copper, zinc, and lead are good conductors, while normal igneous and sedimentary rocks are not: but, water being a good conductor, those rocks containing much water are difficult to distinguish from ores. These methods are used mostly for metalliferous deposits down to, say, 400 or 500 ft. They may also be used for determining the dip of rocks, thickness of strata, the presence of faults, and for confirming information from bore holes.

The Seismograph Method. The speed at which a shock travels through the ground varies with the density of the material. This variation is from 2000 ft/sec in sandy soil to 20,000 ft/sec in granite; in limestone it is five times as fast as in sandstone. An artificial earthquake is made by exploding a charge of high explosive in a hole 20 ft deep, and measuring the speed with which the shock travels through the ground. This is done by noting the arrival of the shock at various points where records are made by seismographs. The instruments are placed up to ten miles apart. Although this method can give no direct indication of oil, it does give most valuable information as to the structural conditions. It has located salt domes on the flanks or crown of which oil pools are found.

The Radio-activity Method. This method depends on the intensity of emissions of radio-active substances, which are determined with an electroscope (Lutz & Wulf electrometer). Chiefly used in determining fault planes, which are channels along which such emissions have a tendency to circulate; but it is only effective at very shallow depths.

Boring

Boring is mostly done for exploratory purposes, such as when testing deep alluvial; determining the depth at which coal seams, iron-ore beds, etc., are to be found at a certain point; the thickness of beds; and the nature of the rocks to be passed through when sinking a shaft. When boring for exploratory purposes, one must be very careful interpreting results, especially in faulted country. Boring is not satisfactory for patchy deposits, or for those with a steep underlie. Negative results also have a value by informing one where not to extend workings. When mining coal under a body of water, bore holes are put down to determine the depth of cover. They are also used for tapping water, gas, or oil, and for pumping up brine or even molten sulphur. Boreholes are sometimes used as a passage for steam or water pipes, or electric wires; and at times for communicating with entombed miners and supplying them with fresh air and food. Boring is a cheaper and quicker way of obtaining information than shaft sinking, as one can bore several holes for the same cost as one shaft, thereby obtaining a better average value of a deposit.

The diameter of a hole depends on the purpose for which the hole is bored, the class of rock to be bored through, and the depth. If boring is through alluvial to obtain samples, the hole is generally 4 to 6 in. in diameter. For a deep hole, say, 1000 ft or deeper, the hole is made larger at the surface than what is required at the bottom so as to allow for contraction in depth when casing has to be inserted.

When sinking an important deep hole, it is better and usual to start two at the same time, so that in case of a serious accident to one, the desired information may be obtained from the other without loss of time. A company seldom has so much boring to be done as to warrant the purchase of a boring plant. A plant may be hired and worked on wages; but it is better to have the work carried out on contract by experienced men.

For boring purposes, rocks are classified as—

(*a*) Hard, such as granite, diorite, basalt, and metamorphic rocks.

(*b*) Medium, such as shales, slate, and soft sandstones.

(*c*) Soft, such as sand, gravel, clay, and decomposed rocks.

Boring may be done by percussion or rotation. Percussion is best for fairly hard and porous rocks, like some kinds of basalt, which would break off the diamonds of a diamond drill, or lose shot of a shot drill. The rotating type, such as diamond and shot drills, are suitable for hard uniform rocks. Augers are used for soft rocks. Holes, unless in hard rock, have to be cased to prevent the sides from falling in.

Hand Boring. At first, the men may use a capstan head with which to lift and turn the rods, but when the depth exceeds 100 ft, the weight of the tools and rods, together with fraction and partial vacuum, makes 100 ft about the limit for hand work, after which a spring-pole lever, 10 to 12 ft long, or a light walking-beam with a balance weight at the farther end, is brought into use. The ratio of the longer to the shorter arm is 4 to 1. For still greater depths a windlass and tripod are used. It is found that when rods exceed 350 ft they are liable to break on account of excessive vibration, so rope boring is substituted.

The rods are generally 1 in. square section, made in various lengths from 12 in. up to 10 ft. The longer the rod, the less unscrewing there is to be done, but they must not be too heavy and awkward to handle. The tiller is best clamped on the top rod, so that it can be easily adjusted. The male screw is on the upper end of the rod, for, if the female screw was on top, dirt might fall into the cavity and prevent a tight joint from being formed.

The tools mostly required (see Figs. 24A and 24B) are—

Hand-dogs (*j*) for screwing and unscrewing the rods, the end being pointed so that it can be used for tightening the tiller screws.

A swivel top rod (*a*) to which a rope is attached for raising the rods.

A split-hook (*k*) to slip under the swelled joint of a rod, whereby it can be raised.

A safety wrench to hold the rods back from falling into the hole when a length is being unscrewed.

A pipe clamp to prevent the casing from falling back into the hole when it is being pulled out.

A tiller (*e*) made so that it can be folded up when not in use.

Augers of various types according to the class of work they are required to do. The worm-auger (*d*) is used for loosening material; the strong-auger is required for stiff clay or marl; the parallel-auger for bringing up stiff clay and loosening tight, sandy, and gravelly soil; the open-auger (*f*), closed at the top and with a cutting edge at the bottom; the shell-auger (*g*), open at the top to allow air to escape; the bottom has a cutting edge and a valve to prevent loose material from escaping.

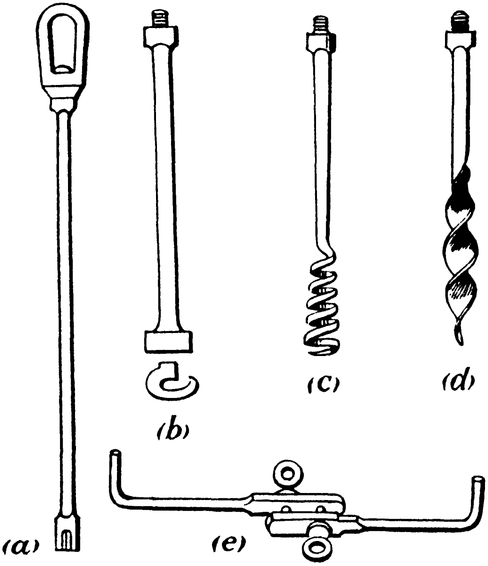


FIG. 24A. BORING TOOLS

(*a*) Swivel top rod; (*b*) Crow's foot; (*c*) Spiral worm; (*d*) Worm auger; (*e*) Folding tiller

An auger cleaner.

Chisels with different cutting edges, straight (*h*) for ordinary purposes, V-shaped (*i*) for hard ground.
A sand-pump or sludger.

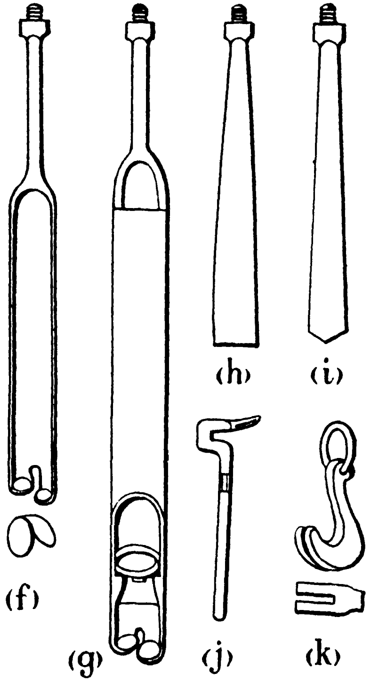


FIG. 24B. BORING TOOLS (*cont.*)
(*f*) Open auger; (*g*) Shell auger; (*h*) Flat chisel; (*i*) V-chisel; (*j*) Hand dog; (*k*) Lifting dog

Various fishing tools depending on where the break has taken place. The most common is the crow's-foot (*b*) for gripping a rod that has become unscrewed; it is worked round the rod and drawn up under the shoulder of a joint. The bell screw is a conical female screw which fits over the male screw of a rod that has become detached. The spiral-worm (*c*) is used for extracting rods when the shoulder of a joint has been broken off. The ram's-horn is a double spiral, wider at the bottom than at the top, used for grappling a broken rope.

In the best practice the casing is set a foot ahead of the drill. The casing is lowered by its own weight, together with that of a platform which is attached to the top of it and the men working the drill; at the same time the casing is rotated by men or beast on the ground. The casing is withdrawn by the use of a lever, being rotated all the while. It is usually withdrawn at the rate of 30 ft or more per hour. The casing being in lengths of 5 ft is easily handled. The sections are screwed together with flush joints; the threads are small V-threads, square threads have not proved successful. Casing 4 in. external diameter is $\frac{1}{4}$ in. thick; casing 6 in. diameter is $\frac{5}{16}$ in. thick. The bottom length of casing has a driving shoe at its end. The spring dart is an instrument somewhat like a harpoon for drawing casing. The wings are pressed in while in the pipe, but as soon as the instrument passes outside the casing the wings fly out underneath so as to hold it.

For shallow depths, down to 20 ft, a sample may be obtained through clay, sand, or fine gravel by driving a 6 in. pipe through it and about 2 ft into the bed rock to serve as a plug. On withdrawing the pipe, its contents are carefully cleaned out and the different layers panned off. The pipe is driven with a pile driver weighing about 500 lb and given a drop of 2 ft. The chief objection to this method is the fear that the pipe might be withdrawn before the bottom of the deposit is reached; but an experienced man can generally tell by the sound.

Rope Boring. Deep sinking with iron rods has given place to rope boring, where a rope takes the place of rods. It was found that a string of rods and tools 400 to 500 ft absorbed about 95 per cent of the energy by the elasticity of the rods, and the steel had a tendency to crystallize, particularly where a rod screws into its coupling. An effort to overcome this was made by inventing a free-falling device, which consisted of an apparatus at the end of the string of rods which picked up the cutting tool and then, when drawn up a certain height, allowed it to drop. A flexible rope is more readily handled than iron rods for deep boring, and it greatly reduces the time and trouble required in raising and lowering heavy rods. Only vertical holes can be sunk by rope boring. Some plants are portable, but for very deep holes a separate derrick, 30 to 50 ft high, is erected.

Two ropes are employed: one for drilling, the other for the sand-pump. Their sizes are—

	Diam. for Drill Rope	Diam. for Sand-pump
For depths less than 300 ft .	$1\frac{3}{8}$ in.	$\frac{5}{8}$ in.
From 300 to 500 ft . .	$1\frac{1}{2}$ in.	$\frac{7}{8}$ in.
From 500 to 800 ft . .	$1\frac{5}{8}$ in.	$\frac{3}{4}$ in.
From 1000 ft to 2000 ft .	$1\frac{3}{4}$ in.	$\frac{3}{4}$ in.
For 2000 ft or over . .	$1\frac{7}{8}$ in.	$\frac{7}{8}$ in.

The drill rope is wound on an 11 ft diam. bull-wheel, and passes from that over a crown pulley at the top of the derrick and thence to the temper screw, where it is gripped. The temper screw is attached to one end of a working beam. The function of the temper screw is to enable the tools to be lowered a little at a time when the hole becomes deeper without having to let the rope off the bull-wheel so frequently (Fig. 25).

A full set of tools consists of a rope-socket, sinker-bar or stem, jars, drill-bar, and bit. They are 30 to 40 ft long and weigh from 800 to 1600 lb, according to their diameter and length. The process of drilling till the whole set of tools is connected is slow. For shallow holes, the string of tools consists only of the rope-socket at the top, the stem, and immediately below that of the bit. Jars need not be used till the hole is down 50 ft, and under favourable conditions may not be required at all. The rope-socket connects the rope with the sinker-bar. The sinker-bar is about 18 ft long, and keeps the rope taut and the upper part of the jars in line. The jars are made in two parts and are like long links of a chain. When extended they are 6 ft long and, when closed, 5 ft 3 in., the play being 9 in. The object of the jars is to loosen the bit when stuck by giving an upward blow. Sometimes the upper part of the jars is allowed to strike the lower part to procure a double blow when drilling through very hard rock. The drill-bar, like the sinker-bar, is a plain iron rod provided with screws for connecting together, their total length ranging from 6 to 45 ft. The tools are raised from 18 to 36 in. according to circumstances and are then

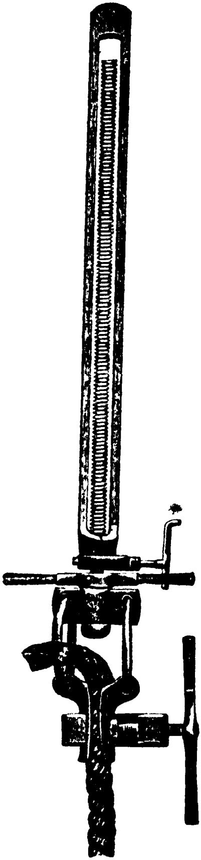


FIG. 25. TEMPER SCREW

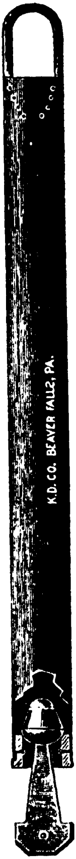


FIG. 26. SAND PUMP WITH DART

allowed to fall their full weight. They strike 18 to 20 blows per minute. An experienced driller can tell the nature and thickness of the rock passed through by the feel of the rope, the difference in the rate of cutting, and by examining the material brought up by the sand-pump. The operator turns the drill about an eighth of a revolution in the same direction after each blow for 30 or 40 strokes, and then reverses the twist of the rope. If the rock will hold water, about 2 gal of water are used for each foot drilled, so as to float the cuttings off the bottom. A hole is started with a spudding-bit, which is thinner and sharper than the regular bit. If the rock is both hard and full of cracks, or if it stands on edge, a paddle-bit is used which is nearly as wide as the top of the cutting edge and the steel, being thick, nearly fills the hole, so it cannot slip off sideways. All joints must be screwed up tightly. Half the fishing troubles are due to the joints not being screwed up. Marks should be made on adjoining pieces, so that one can tell if they have been properly screwed up.

The sand-pump (Fig. 26) is a wrought-iron cylinder 6 to 8 ft long, with a valve at the bottom and a strap at the top to which the sand-rope is attached. The sand-rope passes over its special crown-wheel and is worked by friction. When the sand-pump is lowered to the proper depth, it is churned up and down two or three times and then drawn to the surface. Sometimes a small bit or dart is attached to the bottom of the valve to loosen any dirt at the bottom of the hole, so that it can be drawn into the shell. A vacuum pump may be used instead of a common sand-pump. but the latter being lighter and easier to handle is generally preferred. The great advantage of a suction or vacuum pump is that it will draw into itself everything small and loose enough to be taken, while the common sand-pump will only take what will pass through the valve without other assistance; so when the point is reached where valuable minerals are expected a vacuum pump should be employed.

The top of a hole is lined with casing to keep the sides from falling in and, when necessary, other parts of the hole are also lined. Unless casing is used from the surface down, each successive section has to be smaller in diameter, necessitating the use of smaller bits. The casing must be forced down through all obstructions quite vertically, which requires great skill. An iron driving-cap is screwed on to the top of the casing and driven down with a monkey or ram of wood 6 to 8 ft long and 12 to 14 in. square section, which is bound with iron bands at the battering end and is provided with a hook at the other end. The bottom of the casing has a bevelled cutting edge. If hard rock is met with, a hole is drilled two-thirds the outside diameter of the casing; the casing is then driven down to break off the edges of the obstacle, which fall into the space bored out by the bit.

When a bore is finished and the hole is no longer required, as much of the casing as possible is recovered. There are various ways of doing this. One is to lower an olive-shaped piece of iron that nearly fits the casing, and then drop sand and fine gravel on the top so as to cause it to jamb and get a firm hold; or screw-jacks may be so arranged as to draw the casing up. A casing elevator consisting of a hinged clamp is fixed round the casing below a coupling: the two links attached to the clamps are suspended from a swivel-hook so that the casing can be turned as it is being lifted. If the casing is stuck, it may have to be jarred free. To do this, the sinker-bar is placed inside the top of the casing and suspended by a rope passing through a hole in the drive head, which is screwed on to the casing. The sinker-bar is then lowered a certain distance and brought up with a jerk, hitting the cap with considerable force. Bits wear out of gauge quickly when working in hard rock, necessitating a reduction in the diameter of the hole, or else the expense of reaming it out for fresh bits of the original size. There are various forms of reamers.

Accidents occasionally happen. The drill may become dislodged, or it may become wedged, when it must be persistently jarred to release it. The sand-pump may break away from its rope so that it has to be fished up. A piece of a tool may break off; if small, it may be brought up by a magnet. If tools becomes wedged by a rush of mud which sets round them, it may be necessary to cut them out with a spear-pointed instrument attached to a light set of rods. If the bit gets broken and cannot be fished up, the borehole may be filled for about 100 ft with some hard rock or cement, and then commence drilling with a view to diverting the hole so as to drill past the broken bit. If a soft patch or hard rock occurs at an angle, the tools may become diverted. To remedy this, a charge of dynamite may be lowered into the hole and fired, which will destroy the irregularity and allow the tool to resume its perpendicular course.

The Calyx Drill. This drill is most suitable for soft rocks and has given satisfaction in Victoria, but does not seem to be much favoured in other countries. It is a rotary drill which yields a core. It differs from the diamond drill by having steel teeth instead of diamonds. The crown is made of the best steel 10 to 12 in. long and about $\frac{3}{16}$ in. thick. The teeth (Fig. 27) are nearly vertical in front, but at the back are bevelled at an angle of about 60° from the horizontal. The teeth, which are long and sharp, are set in and out, like the teeth of a cross-cut saw, to provide clearance for the core and a passage-way for water. They are readily sharpened on an anvil, and when worn down, the teeth can be lengthened by nipping out from the groove left for water to travel along. The pressure on the teeth, which varies from 500 to 3000 lb, causes the teeth to bury their points in the rock, where they remain stationary till the torsion is sufficient to wedge away particles of rock in front of them. The motion is a series of jerks rather than a grinding action., The rods are hollow and water passes down them. The core-barrel is attached to the bit, and above that is a long sediment tube known as the calyx, almost the same diameter as the hole. Water carries chips and fine material up between the side of the hole and the calyx, but when the calyx ends and the current is reduced, the coarser particles fall into it. The drill is given about 12 r.p.m. The core is broken off by dropping a few handfuls of grit down the hollow rods which pack between the core and the barrel; the tools are then pulled out.

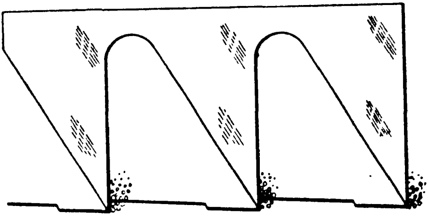
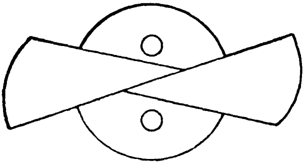
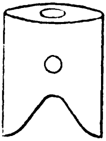


FIG. 27. TEETH OF CALYX DRILL CUTTER

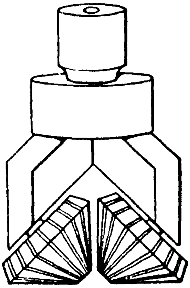
Rotary Attrition Drills. These drills are rapidly replacing others for sinking oil wells where the rock is not too hard. Water is passed down the hollow rods and through two holes in the bit. One can drill without casing by using muddy water under pressure until the drilling is completed, so the size of the hole is the same from top to bottom. A horizontal table, driven by gearing which rotates on cone bearings, is perfectly enclosed and revolves in a bath of oil. In the centre of the table is a hole through which the drill-stem passes and is forced to rotate with the table. The lower end of the drill-stem is attached to the drill rods; the upper end to the water swivel which carries the weight of the drill rods and tools. The bits are made $\frac{1}{2}$ in. smaller than the casing in which they have to work if the hole is lined. The bits (Fig. 28 (*a*) (*b*) (*c*)) are given various shapes, but are generally fish-tail, drag, diamond, saw-tooth, cone, cross-roller, or disc. The fish-tail is satisfactory for soft formations, and under ordinary conditions will last 24 hours before it requires re-sharpening. When a rock is reached that cannot be negotiated with a fish-tail, one of the roller bits is used. These bits have serrated edges which, under pressure, crush the rock face. To a depth of 3000 ft rods of 6 in. diameter are generally used, but for a greater depth, on account of their weight, 4 in. rods are employed. Derricks 130 ft high allow 4 rods, totalling 96 ft, to be pulled at a time. Cuttings are removed automatically by the flow of water, so there is no occasion to stop and change to a sludger. If water is lost in a stratum, the driller immediately thickens his mud so as to plaster up the sand. When boring for oil or gas, an initial column is generally set for a few hundred feet and cemented in, in case of an unexpected blow out.



(*a*) Drag



(*b*) Fish tail



(*c*) Cone

FIG. 28. BITS

Diamond Drilling. Diamond drills have an advantage over other types, inasmuch as holes can be drilled in any direction, but as they do not always go where intended, deep holes should be surveyed from time to time to avoid expensive mistakes. They are made in different sizes, and can be used at the surface or underground.

The tools consist of a bit or crown at the bottom, followed by the core-barrel, hollow rods, and the feed mechanism. The core-drill cuts an annular groove instead of pounding the whole rock like a percussion drill. A steady stream of water is forced down the inside of the rods by a pump and, issuing at the bottom, rises to the surface between the outside of the rods and the inside of the casing or wall of the borehole, carrying with it the ground-up particles of rock; the water also helps to keep the bit cool. If the rock bored through is soluble, such as rock salt, a saturated solution of the same material is used instead of fresh water. As the boring proceeds downwards, the core of rock passes into the core barrel, where it is gripped by a core lifter, which is a tapered split ring provided with lugs which fit into a conical recess while drilling is going on; but when the core is lifted, it is forced into a narrow part. The core is broken off from time to time by pulling.

The derrick is from 40 to 45 ft high. The boiler is generally 12 h.p. and there is a force pump.

The rods are generally lap-welded wrought iron or seamless steel tubing connected with flush joints, the sections being 5, 10, or 20 ft from joint to joint. As the rods become longer and heavier, they are counterbalanced, so that an even pressure shall be maintained on the bit.

The core-barrel has the same outside diameter as that of the bit, but the inside diameter is slightly larger, and it may be in one or more lengths. When boring through soft material, such as coal which might be disintegrated by the water forced down the tube, a double core barrel is used, the inner one not turning with the rods. As the core barrel almost fits the hole bored, grooves are cut on the outside to allow the uprising water and debris to escape.

The water swivel forms a connection between the top of the rods and a flexible pipe leading to the pump, but it is always leaking.

The bit is of mild steel. There is no point in making it too long, as it would wear in diameter and become too small for the hole before it became too short. There are two classes of diamonds used. These are borts, and carbons or black diamonds, and they vary in size from ⅛ to 5 carats according to the size of the bit. Carbons are preferred as being tougher.

They are more expensive than borts. Borts are real diamonds, which, owing to imperfections, cannot be used as gems. They are employed for medium rocks on account of their cheapness, but owing to their cleavage cannot be used for hard rocks. The wear of diamonds while drilling is very small, but they are apt to become broken. The average life of a setting, taking accidents into consideration, is 10 to 50 ft, and a bit will stand three settings. It is false economy to work too long with one bit, as there may be insufficient iron to grip the diamonds. Small diamonds are not economical, as when they wear down to half a carat, they cannot be reset. Carbons of 1 to 1½ carat can do the same work and are much cheaper than stones of 3½ carat; they are easier to set, and in case of loss not so expensive. Those carbons having straight edges, with sides forming an obtuse angle of 95 to 140°, are the most durable. Reject carbons having a cokey structure, also those having thin sharp edges. If a diamond becomes polished during drilling, turn it so as to give a fresh cutting face. The diamonds should be allowed to project $\frac{1}{64}$ in. from the bit, and should be carefully gauged.

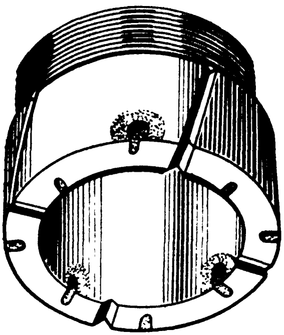


FIG. 29. DIAMOND BIT, SHOWING HOW DIAMONDS ARE SET

In general, a 2 in. core is found to be the most satisfactory size. For bits 1½ to 2 in. diam., one can use ⅝ to ⅞ carat stones. As a rule, a 2 in. bit requires 6 diamonds; a 3 in. bit, 9 diamonds; a 4 in. bit, 12 diamonds; and a 5 or 6 in. bit, 15 diamonds. The stones are so placed that each has its share of cutting to do and does not cut in the groove of the one ahead (Fig. 29). Sometimes sapphires of bad colour are coated with graphite and sold as carbons to those who do not know any better, thereby accomplishing a double fraud, the sp. gv. being 4 against 3·5, and the hardness being 9 against 10. When drilling in rocks of the hardness of limestone, one can reckon on 100 ft per setting. In soft rock one may drill 300 to 500 ft per setting. The speed of the drill is 150 to 200 r.p.m.

Non-core bits economize drilling time, as there is no stopping to draw the core, since the worn material passes out with the water. Diamond drilling underground makes much less dust than percussive drilling. For machine drilling, 15 to 20 ft is considered fairly deep, but diamond drill holes have been made 100 ft with ease.

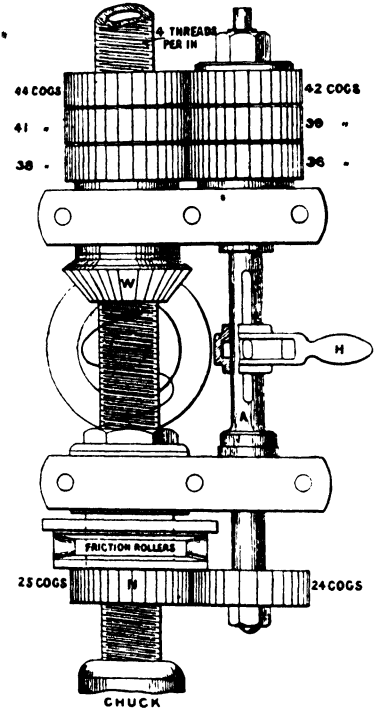


Fig. 30. Positive Feed

For boring in hard ground, whole diamonds or borts ground to 40 mesh are sometimes mixed with powdered metal (a mixture of copper, cobalt, and molybdenum is used), placed in a mould and heated in an electric furnace to about

1000° C, then placed in an hydraulic press. When cold, it is taken from the mould, cleaned, and silver-soldered to the shank.

When running normally, the core is a smooth cylinder. Vibrations cause irregularities from roughness to a corkscrew pattern.

A drill may be diverted from its course by various causes, such as starting a smaller diameter hole without properly centring the bit; by having too great a weight on the bit, which bends the rod to the limit of the diameter of the casing; by failing to take into consideration the varying degrees of hardness of the different strata when lying at an angle; by failure of the pump to carry away the cuttings, resulting in a tendency of the cuttings to crowd the bit.

The two main methods of feeding the drill downward are the positive screw feed and the hydraulic feed. The positive feed (Fig. 30) is purely mechanical, and is essentially an arrangement of differential gearing so proportioned and adjustable that varying useful rates of feed for different hardness of rock may be obtained. This method is suitable for uniform drilling conditions and is used on the smaller machines. The gear may be made to descend 1 in. for every 309, 433, or 714 revolutions, which in practice are found most convenient. With Bullock's mechanism, a hollow spindle 5 to 7 ft long, with four threads per inch, passes freely through the bevel-wheel (*W*) driven by the mitre-wheel on the engine shaft, but can be keyed to it by means of two feathers, which fit in keyways running the whole length of the spindle. (*N*) is a bottom spur-wheel, the boss of which forms a nut fitting on the quill. If both nut and screw revolve at the same speed, there will be no travel of the screw; but if (*N*) be revolved more quickly, the spindle will have a downward movement and carry the bore rods with it. If the nut is held stationary and the screw has four threads to an inch, then for every four revolutions the rod would advance 1 in., which would be much too fast. The feed is obtained by varying the relative speed of rotation of the hollow spindle carrying the boring rod and the bottom spur-wheel (*N*), thus—

$$\frac{\text{Driver } 44 \times 24}{\text{Driven } 42 \times 25} = 1{\cdot}0057 - 1$$

One is deducted as each wheel turns round once for each revolution, making it 0·0057. The screw shaft has 4 threads per inch, so—

$$\frac{4}{0{\cdot}0057} = \underline{\underline{714\text{ r.p.m.}}}\text{ advance.}$$

$$\frac{41 \times 24}{39 \times 25} = 1{\cdot}00923 - 1$$

$$\frac{4}{0{\cdot}00923} = \underline{\underline{433\text{ r.p.m.}}}$$

$$\frac{38 \times 24}{36 \times 25} = 1{\cdot}0133 - 1$$

$$\frac{4}{0{\cdot}0133} = \underline{\underline{309\text{ r.p.m.}}}$$

The boss of the wheel (*W*) is prolonged above the bearing, and carries three spur-wheels which rotate with it. These wheels gear with three other wheels, which run loose on the countershaft (*A*). This countershaft is hollow, and by means of the handle (*H*) a key can be raised or lowered to engage with any of the three wheels, which will then cause the countershaft to revolve. At the bottom of the countershaft is another spur-wheel which engages that of (*N*).

The hydraulic feed (Fig. 31) is preferred for large drills. It is quicker than the positive feed, causes less wear and tear of diamonds, there is less strain on the boring rods, and it is not limited to four speeds. With this apparatus the pressure on the diamond bit is constant, the speed varying as the bit passes through soft or hard bands; at the same time any change in the stratum is registered by the hydraulic pressure gauge attached to the cylinder and, if necessary, the feed can be altered. The feed may be single or double.

When two cylinders are used, they are connected with a cross-head. To the upper end of the piston rod (*C*) is screwed the thrust-plate (*G*) through which pass three studs screwed into another thrust plate (*H*). Between these thrust-plates are two sets of friction ball-bearings, one set on each side of the collar (*I*), which is screwed on to the rod (*J*). One set maintains the weight of the rods when they hang in the drill clutch; the other sustains the upward thrust of the rods when drilling. The collar transmits the vertical motion of the hydraulic piston to the drilling bit, for as the piston and piston rod descend, they carry with them the two thrust-plates (*G* and *H*) and the two roller bearings with the collar between them. The collar rotates with the drive rod, which is rotated by the mitre gear (*K*) through which it slides, the

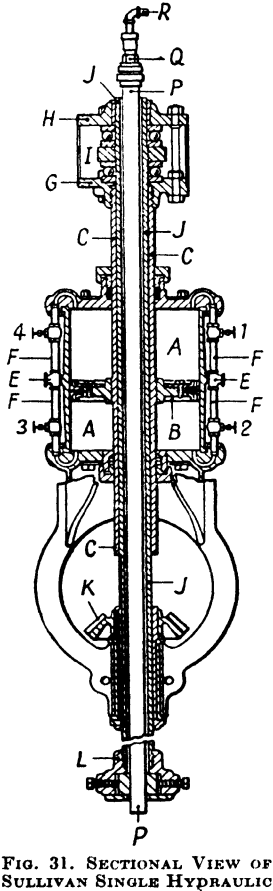


Fig. 31. Sectional View of Sullivan Single Hydraulic Feed Mechanism

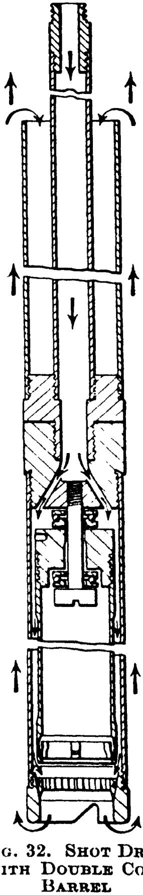


Fig. 32. Shot Drill with Double Core Barrel

feather of the mitre gear sliding in grooves in the drive rod. Thus the piston rod and drive rod descend together, the latter rotating within the former. The bore rod (*P*) passes through the drive rod (*J*), and is held and rotated by means of the chuck (*L*) screwed to the bottom of the drive rod. A high-pressure pump is connected with the cylinders at a tee, the water passing into the cylinder (*A*) through the brass tubes (*F*) and the ports cut in the cylinder heads. The water valves 1 and 2 are the inlet valves, and 3 and 4 the outlet valves. When in operation, water is pumped through valves 1 and 3, which are opened, while 2 and 4 are closed. When the water is above the piston (*B*), its downward movement is regulated by the valve on the escape tee. If 2 and 4 are open, and 1 and 3 closed, the water pressure is below the piston which consequently rises. But little water is used as it is employed over and over again; water for drilling purposes passes through the pipe (*R*) and the water swivel (*Q*).

Should the rods show a tendency to stick in the hole while the machinery is running, the feed can be stopped without stopping the revolutions of the bore rods as must be done with the positive feed. The pressure upon the piston in the water cylinder is generally kept at 80 lb/sq in. Too great a pressure tends to dislodge the diamonds from the bit. The pressure exerted by the feed should be just enough to produce abrasion. If a diamond breaks loose, it may be recovered with putty stuck in a hollowed out piece of wood inserted in the end of the drill rods, or a small hole may be drilled at the bottom of the bore into which the diamond is washed and then the hole is continued its full size, the diamond being brought up with the core. If a bit becomes detached at the bottom of a hole, it may have to be chopped in pieces, and the pieces brought up by a powerful magnet.

The Shot Drill. This (Fig. 32) is a modification of the calyx drill, suitable for hard rocks of nearly uniform texture without seams and crevices, but for vertical holes only. It is not suitable for soft rocks in which the shot may become embedded. The bit is a plane cylinder of steel with a slot in it near the bottom. Chilled shot made by atomizing high carbon molten steel is fed down the hollow drill rod and does the wearing away of an annular groove by the weight and rotation of the string of tools. A sphere touches a flat surface at a point, so the numerous shot act as so many minute pick points. The average size of the drilling shot is about $\frac{3}{32}$ in. in diameter. Of course the bit is worn as well as the rock, but not at the same rate, and the bit is not very expensive to replace. The amount of shot required naturally varies with the rock; shale, slate, limestone and ordinary sandstone may require from ¼ to ¾ lb shot per foot of hole; very hard sandstone, granite, quartzite, and such-like rocks require from 1½ to 4 lb/ft. In most rocks the shot automatically cuts the necessary clearance inside the bit so there is a space between the core and bit for the shot to pass to the bottom of the hole, but to be on the safe side two grooves are milled lengthwise inside the bit. The drill revolves at the rate of 80 to 100 times per min. If the rock is soft a double core barrel is used. Drilling with chilled shot saves buying expensive diamonds and having a skilled diamond setter.

Kind–Chaudron Process. This method has been used for sinking shafts up to nearly 16 ft diameter through quicksands, which have been lined and pumped out without a man having been inside it. This method is generally used as a last resort and mostly for depths of 300 to 400 ft, though they have been deeper. The boring is done in two sections by instruments known as trepans. The shaft is started by the smaller one, 4½ ft across, which is kept ahead of the larger one made the full size of the shaft.

Sampling

The object of sampling is to obtain a representation of a body of ore in all respects, except bulk. The sampling may be done to guide the mine manager, or a prospective buyer. It is assumed that the mine has not been salted or dressed and that the sampler understands what appliances are required and how to use them. It may be necessary to sample hard ore in situ; alluvial; loose heaps, such as tailings or piles of ore: ore already bagged ready for transport; products of treatment; or slag. Bulk samples are not much good unless required to ascertain the best method of treating the ore, in which case the sample must be representative of the mine. Chip samples from drill holes are not reliable, since much fine stuff (which is often the richest) is lost in dust or sludge. Grab samples are worse than useless.

The results from sampling and that of actual mining seldom agree. This is due to miners not breaking the ore as clean as the sample; there is also loss in handling, loss in float material, in tailings or slag, and the sampler may have used an unsatisfactory method of sampling or calculating. For these reasons it becomes necessary to make certain allowances so that the manager shall have a better idea of what he may fairly expect to save. After a time, a factor may be obtained which will bring the sampling and practical results more in conformity one with the other.

Sampling a Lode. Cuts should be taken from wall to wall. The place where the cut is to be taken must first be cleaned and made level by chipping with hammer and gad or moil, or better still with a jack hammer if available. The sample is then taken by cutting the channel deeper. Each sample should be taken the same width and depth, and should weigh about 5 lb. In a wide lode several consecutive cuts are made in a continuous line each about 5 ft long, unless there are bands of mullock or different kinds of ore which can be kept separate when mining; in which case they are sampled separately. If circumstances permit, classify the lode material according to the way it will be treated (ore rich enough to be treated; ore requiring dressing; and waste). If for any reason samples cannot be taken at regular distances, but the lode is known to exist, allowance will have to be made, generally by averaging samples on either side of the place that cannot be sampled. An occasional high assay may necessitate the taking of another sample in its stead, but if that cannot be done, take the mean of the two nearest cuts. When using a diamond drill for sampling (some minerals are softer than others and are liable to make more sludge), both core and sludge should be assayed together; for if assayed separately, they will not agree. A sludge box is 4 to 6 ft long, 18 in. wide and 10 in. deep, divided transversely by a baffle plate, the upper edge of which is about an inch below the top of the box. Sampling must be done systematically: cuts should not be more than 5 ft apart along the length of the lode if narrow, or more than 10 ft in a wide lode. The nearer the cuts are taken, the more accurate the results.

Sampling Alluvial. Old river beds are generally sampled with a drilling plant since several holes can be put down at the same cost as one shaft, thereby distributing the samples better; the contour of the bed rock is fairly accurately obtained; and water is not so troublesome. The engineer is not concerned so much about the actual gold or tin contained in the alluvial, as to what a dredge will recover, and what layers of clay are likely to be encountered. A convenient scale on which to plot the bore is 2 chains to an inch or, if the area is very large, 4 chains to an inch. In a wide valley where the original river may have meandered about, scout holes should be close enough together (say 8 chains apart) to determine whether a narrow deposit of sufficient richness exists: but if the boundaries of the deposit are known, scout bores may be dispensed with. Normal holes are put down at 4 chain intervals. If the borings agree within 5 per cent, there is no object in boring holes closer: but if the values of the normal holes are not reliable, check bores are sunk at 2 chain intervals.

In exceptional cases it may be necessary to bore at 1 chain intervals. The holes must be lined with casing, generally 6 in. or 4 in. inside diameter. The material brought up from a borehole is placed in a measuring box, 12 in. by 12 in. by 4½ in. marked off vertically inside with brass sprigs in tenths of a cubic foot. The sample is drained, and the bulk obtained by filling up to the mark with a known quantity of water. Samples from boreholes do not show the actual content of the casing because the wall of the casing displaces some of the alluvial; so a correction has to be made. Three constants are in use. The constant is a figure which multiplied by the length of the casing driven will represent the cubic feet of material coming from the casing. One constant having a value of 0·3068 is based on a diameter equal to that of the outside of the cutting shoe (7½ in. in the case of a 6 in. inside diameter pipe). Another has a value of 0·333 calculated to discount the drill hole values automatically. The third, known as Radford's factor, has a value of 0·27 based on experimental research which represents one cubic yard of material to every 100 ft of a 6 in. pipe: this is the one most generally used. Boring results for alluvial are nearly always higher in value than the actual results, as a dredge does not always clean up the bottom, and there is always a certain amount of loss in handling. Some engineers reduce their original figures by 10 to 20 per cent, not because they doubt their figures, but because they wish to allow for reduced recovery.

Sampling a Heap of Loose Material. A dump of broken ore or tailings is by no means uniform; it comes from different parts of a mine, and what looks like ore may only be a cover for a heap of mullock. So-called grab samples are no samples at all. The larger pieces fall to the foot of the heap and are often poorer than the finer material. The only thing is to make one or more cuts through a heap of ore according to its size. The way to sample heaps of tailings is to divide them into squares of convenient size and take vertical samples to the bottom with an auger or tailings sampler: the latter consists of a wrought-iron pipe about 1½ in. diameter of sufficient length which is driven through the tailings, and when withdrawn cleaned out. Unless there is some special reason, all samples from one heap are mixed together, and reduced in bulk by quartering. In estimating the bulk of tailings, the contour of the ground must be taken into consideration.

Sampling Ore in Bags. If the ore is in bags ready for shipment, one bag in so many is taken, depending on the nature of the ore. If very rich, or the values are irregular, it may be necessary to sample the lot.

Sampling Products from Various Machines in an Ore Dressing Plant. This is generally done by some automatic machine.

Sampling Slag. A long iron bar is dipped in the molten slag as it pours into the slag bogie and is then plunged into cold water. A sample thus taken is easily decomposed in strong hydrochloric acid.

Sampling Bullion. Pigs of lead, bars of copper, ingots of gold or silver have samples taken by drilling holes with metal drills.

As the precious metals in lead are not uniformly distributed, the pigs should be laid out so that drillings can be taken from various parts.

Determining the Value of a Block of Ore

There are four methods used for determining the value of a block of ore in situ.

The Arithmetic Mean. The sum of the values and the sum of the widths of the places sampled are each divided by the total number of cuts taken. The result is inaccurate since no allowance is made for the *relative* widths and values, so this method should never be used.

The Geometric Mean. The assay-widths (i.e. feet per cent, feet-ounces, or whatever units are employed) are added together, and the sum of these is divided by the sum of the widths. This method assumes that the width and value are the same for half the distance on each side of a cut, and that the values and widths rise and fall from point to point in a straight line (which is seldom if ever the case in Nature), but if the cuts are sufficiently close together, a fair approximation to the natural curve is obtained.

The Prismoidal Method. This method requires tedious calculations. The area between two cuts is obtained by adding the widths of the two ends to four times the middle area, and dividing by six. The assay value is obtained in a similar manner.

The Graphic Method. This method is superior to any of the others. It was first proposed by E. T. M. Garlick. Lay off on squared paper (10 ft to an inch) the distance apart of the various cuts on a horizontal line known as the axis. From these points and at right angles to the axis, lay off the assay-widths on one side, and the widths on the other. Draw connecting lines (straight or curved as required) between adjoining points of each figure, and join the extremities of the assay-widths and width lines to the corresponding extremities of the axis. The scale to which the assay and width diagrams are drawn is important, for if too small, errors in measurement become exaggerated: if too large the diagram becomes cumbersome. When possible it is advisable to use the same scale for both ordinates and abscissae. When this cannot be done, the distance between samples and the widths of sample-cuts may be drawn to a scale of 10 ft to an inch, and the scale of the assay-width lines to 30 ft per cent to 1 in., which is a ratio of 1 to 3. The area of the width figure (area sampled over) and the area of the assay-width figure are then obtained by means of a planimeter, usually set for 1 in. to 100 sq. ft. The assay-width, if it has been reduced to 30 per cent, is then multiplied by three to bring it to its true relation to the area sampled over. One can more nearly approach the natural curve of an area by this method than even by the prismoidal method. The mean value of the area sampled over can be obtained more quickly than by either the geometrical or prismoidal methods. In the case of wide lodes, where one has to depend on occasional cross-cuts to prove widths and values, this method is more likely to be correct than any of the others.

Value of a Block. The average values of each boundary of the block are added together and divided by the sum of the areas in square feet.

The Tonnage. The number of cubic feet in a ton of water is 32. The number of cubic feet of ore divided by 32 and multiplied by its sp. gr. gives the tonnage. One should obtain the sp. gr. of the ore and not make a guess at it, for if incorrect it may make a great difference in values. The sp. gr. of a quartz vein is very different from that of a lead lode. The sp. gr. of a substance is the ratio of the density of the sample to the density of distilled water at the same temperature. The volume is determined by displacement, and the mass by weight. Use metric weights and measures. Place a wide neck sp. gr. bottle on one pan of a balance and a counterpoise on the other. Fill the bottle with distilled water, dry the outside and weigh. Add the dry powdered sample (say 8 g) to the pan with the bottle on it; weigh again to obtain the weight of the dry sample (W). Pour some of the water out of the bottle, place the dry sample in it; shake the bottle to get rid of any bubbles of air; fill up with water and weigh (w). The loss of weight is due to the displacement of water by the sample ($W - w$). The sp. gr. of the sample is $\frac{W}{W - w}$.

If, on examination of a deposit the sampler finds it necessary to make a deduction of, say, 10 per cent from the value of a block bounded by four sides (which is over large and may contain pinches or horses of mullock) for workings with one hidden face he should deduct 15 per cent, for two hidden faces 20 per cent, for three hidden faces 33 per cent—if indeed he should make any estimate when only the outcrop is showing. The only ore in sight is that actually broken. Ore exposed on four sides should be spoken of as "Ore blocked out." Ore which is not exposed on four sides may be termed "Ore which may reasonably be assumed to exist though not actually blocked out."

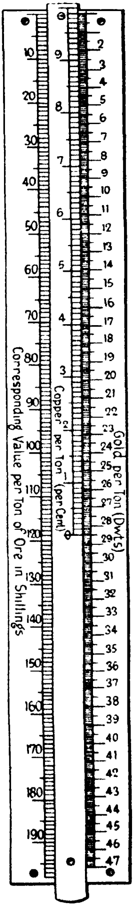


FIG. 33. SLIDE-RULE

Slide-rule for Computing the Value of Ores

A convenient slide-rule (Fig. 33), useful for calculations and for rapidly determining the value of an ore containing two metals, may be constructed as follows—

Take a gold-bearing copper ore as an example. Assuming the gold to be worth 4·24s. per dwt and copper to be worth £60 per ton or 1200 shillings, then one unit is worth 12s. Divide one side of the stationary scale into a convenient number of parts of suitable size to represent pennyweights, say three divisions or pennyweights to an inch, and subdivide each of these into tenths, having zero at the top of the scale. On the other side of the stationary portion of the scale divide into shillings, making every 10s. more distinct. The zero for shillings is also at the top of the scale, and the divisions of the scale are so made that 4·24 correspond to the 1 dwt line. The sliding portion of the scale is now divided into per cents of copper, for as much copper as the ore is ever likely to contain. The zero is towards the bottom end of the scale. Each per cent is made equal to 12s. and is subdivided into tenths.

To use the scale, place the percentage of copper against the pennyweight of gold as shown by assay, and read off the combined value on the money scale.

MINING METHODS

Surface Mining

Hydraulic Sluicing. This method of mining alluvial deposits containing gold or tin is employed in places where there is a considerable height of wash above the natural surface drainage, and without too much overburden. A sufficient quantity of water must be available which can be brought to the mine and delivered at such a height as to give the necessary head or pressure for sluicing. The actual process of hydraulicing is cheap, but the outlay to bring water from below snow line to the mine, which may be for some miles, is often costly. Given sufficient water and a suitable head, there must still be a place where the tailings can be deposited without raising the level of creeks or rivers, or of destroying valuable river flats (for alluvial when loosened occupies about a third more space than when packed by nature). The ditch which brings water to the ground to be worked is called the *head-race*. This being open gives no head or pressure. Water only becomes effective when enclosed in a pipe, and this pressure is called the *hydraulic head.* Each foot in height has a pressure of 0·433 lb/sq in., less loss due to friction. To reduce friction losses, the pressure pipe should be as short and straight as possible; and as friction increases with velocity, the larger the pipe for a given quantity of water, the less the loss from this source. The channel cut in the bed rock along which the gravel is washed is known as a *sluice.* There may be branch sluices leading to the main sluice; and the boxes in which the gold or tin is saved are called *sluice boxes.* The channel at the end of, and formerly occupied by, sluice boxes is termed the *tail race.*

The cross-section of the head-race largely depends on the nature of the ground: if of soil, it is made half a hexagon; if of rock the sides are vertical; if the soil is loose and leaky, it is lined with wood, the sides of which are also vertical. If the head-race is dug in soil, the velocity of the water should not exceed 25 ft/mn. On the other hand, the rate must not be so slow that water weeds will grow and retard the flow of water. In rocky ground, and with wooden flumes, as they are made narrower than a ditch in soil to save expense, they are given a greater grade to carry the same amount of water. However, there must be a limit to this, for fear of losing head at the other end; also when junctioning with an earthen ditch again, that point is liable to be scoured. The usual grade of a head-race is 12 ft/ml. Leaks in earthen ditches are liable to take place owing to fallen trees, rotting roots, landslides, and burrowing animals. If freshwater crayfish cause the trouble, a few bags of burnt lime placed in the ditch above their holes will get rid of them. A stock of slabs is stacked near places where a break away is likely to occur. Short cuts may be taken across gulleys by erecting trestles to carry flumes; or water may be conveyed through pipes suspended from a wire rope; or pipes may be laid down one side of a gulley, along the ground and up the other side, for being enclosed any irregularities in the ground do not matter.

In certain cases it may be necessary to carry a flume on brackets attached to the face of a cliff; or a short tunnel may save much trenching and upkeep, besides saving loss of head. In places along the head-race, usually where it crosses a creek, two gates are placed in the race, one to divert the water when necessary to make repairs lower down, the other as a spillway. The side gate is made the height water in the race is allowed to rise, so that any excess of water can overflow in time of flood instead of wearing away the ditch. Any vegetation in the neighbourhood of trestles supporting a flume should be kept down in case of bush fires.

Where the head-race junctions with the *feed pipe* is a *pressure box* or *penstock*. At the end of this box nearer the head-race is a *sand box* 10 to 30 ft long and 2 ft deep to trap any sand washed down, which is provided with a waste gate through which the sand can be flushed from time to time. An inclined grating of bars is placed near the feed pipe to strain off any sticks and leaves that may have fallen into the water. The pressure box is generally about 8 ft deep as the entrance to the feed pipe must be well covered with water to prevent air from being sucked in which passing through the nozzle would scatter the water and destroy its force. The effective head of water commences where it enters the pipe in the pressure box. For small heads, say 50 ft, the feed pipe may be of heavy duck supported by iron rings 3 in. apart which are connected by cords; but is generally of iron, the gauge of which depends on the diameter of the pipe and the pressure it has to withstand. One or two feed pipes may be supplied from the same pressure box. Six in. brass air valves should be inserted in the pipe every 100 ft in length to prevent it from collapsing through the formation of a vacuum, and also at all high points where air is likely to collect and reduce the area available for water. Blow-off valves should be inserted at all low places along the pipe line to get rid of any solid matter that may collect there. Near the lower end of the line there should be a safety-valve set at a pressure slightly above the maximum due to head as then, should the gate be closed too quickly, the valve will relieve the shock and thus preserve the whole line from being ruined. In places subject to extreme changes of temperature, it is necessary to have a good expansion joint every half mile on all pipe lines. All unavoidable bends along the line should be braced and weighted. The sections of piping are connected together stove-pipe fashion. On inclined sections these are held together by binding wire round hooks riveted near the ends of sections, and by anchoring occasionally to posts driven into the ground to prevent the pipe working down hill. Leaks in new pipes often stop after a time owing to sediment filling them up: they may also be stopped by emptying sawdust or horse dung into the top of the pipe. The pipe line should be filled with water gradually so as to avoid straining it.

The giant nozzle at the end of the pipe is bolted to a platform which is weighted with boulders to steady it. There are vanes in the nozzle to prevent the water from feathering. The nozzle must be easy to handle and capable of being directed vertically or horizontally. A counterweight enables the operator to move the nozzle vertically; and a deflector, which is a short tube pivoted on the end of the nozzle, enables the giant to be moved horizontally. When possible the nozzle should be placed where it cannot be buried by a fall of gravel: but if this cannot be done, the operator should work the nozzle from a distance by electricity. If it is buried, it will have to be washed clear by another nozzle. Since in nature landslips take the form of a horseshoe, the operator should never work within a face of that shape, as ground, especially in wet weather, is liable to break away with little warning. When hydraulicing, two nozzles are generally employed, the orifices of which vary from 3 in. to 9 in. in diameter. The smaller diameter nozzle is used for sending a strong jet of water to undercut the gravel at its base and cause it to fall: the larger diameter nozzle is necessary to wash the gravel into the sluice on its way to the sluice box. The bed rock, when exposed, is carefully cleaned up, and if soft it may pay to take up a few inches of the bottom and put it through the sluice box. The dimensions of a sluice box depend on the quantity of water available, the nature of the material to be removed (being wider for sand than for gravel) and the grade given the box. The water should at least cover the largest stone to be washed down. If, on the other hand, the water is too deep, the current might be strong enough to wash the gold too far down the sluice box.

In order to make the gravel fairly uniform in size, the larger pieces may be removed by sluice forks with 8 or 10 tines; or if there are too many large pieces to be handled this way, when the material has travelled sufficiently far to be thoroughly washed, the water and gravel pass over a grizzly made of iron rails where the larger stones are separated from the finer gravel and sand and fall into a hopper from which they are trucked away, while the fine material is treated in sluice boxes given a grade of from 2 to 12 in. in 12 ft, generally 6 in. If there is clay in the gravel, longer sluice boxes are used so as to give gold a chance to be set free, for clay has a way of picking up gold as it rolls along. If the gravel is cemented, steps or drops are made occasionally in the length of the sluice box to help break it up. The length of the sluice box varies considerably, but as a rule most of the gold is caught in the first 200 ft. Another method of separating coarse alluvial from the fine is to use a pebble or grizzly elevator. This consists of a wooden chute about 90 ft long; the sides of the chute taper from 12 ft at the bottom to 4 ft at the top. It is built on bents at an angle of about 17° from the horizontal. The chute is connected with the bed rock by a 10 ft apron, the sides of which fit inside the elevator chute. The bottom of the chute is solid for the first 20 ft, but consists of a horizontal grizzly for the upper 70 ft. Underneath the grizzly is a false bottom which guides the fines to the sluice box which runs at right angles to it. A wing about 10 ft high is built on either side of the chute, one of which reaches to the face. The nozzle that does the undercutting drives the wash between the wings and about 80 ft back from the centre of the chute. A few cubic yards of the material is forced up the solid portion of the incline, the piper then boils out the fines, and when the heavy stones are clean they are pushed over the end of the elevator.

Riffles serve the double purpose of saving gold and protecting the bottom of the sluice box from wear and tear. There are many different forms of riffles, depending on the class of wash to be treated and the material available. Rock riffles are large hard cobbles of uniform size selected from the wash, placed on end with the upper portion facing slightly up stream. They are laid in sections, each section being separated from its neighbour by wooden battens nailed to the bottom. By this means, if one section is damaged, the area is limited. When used, rock riffles are placed at the head of the box and being rough they help to break up the wash: but they are more expensive to make and take up than other kinds of riffles. Wooden blocks, square or round in cross-section, are laid in rows. The square blocks have a batten placed between each row, but the round blocks, not being so regular, are laid in sections like rock riffles. In both cases the end of the grain forms the wearing surface. The wood must not be too soft, otherwise it will wear away quickly; nor too hard or else it will wear in grooves. When blocks become too worn, they are turned upside down; and when they are worn too thin, they are burnt and their ashes washed for gold. Iron riffles can be made out of old iron rails bolted together to form rafts of a convenient size to handle, and can be placed in the sluice box either lengthways or crossways, the former being necessary if sluice forks are used.

Angle iron may also be bolted together with spacing pieces between each length. The side of the angle iron on the top faces down stream; if it faced up stream, eddies formed between the riffles would not allow the gold to remain there. Another form of riffle is known as the *venetian* which is an iron casting not unlike a venetian blind in appearance. For fine material *curley riffles* are sometimes used. These consist of iron bars $\frac{3}{16}$ in. thick by 3 in. deep which are crimped so as to form spaces 3 in. by 2 in. and are bolted top and bottom.

If the sluicing season is short, the supply of water limited, or the sluicing carried out on a large scale, the sluice boxes are duplicated so that no sluicing time is lost. The clean-up is done in sections of about 100 ft. First the gravel is washed off the riffles, and a small amount of water is allowed to flow through the boxes in which the riffles are washed and then placed on one side. Quicksilver which had been added near the head of the sluice box is scooped up together with any amalgam formed, and placed in enamelled buckets to be treated later; the rest of the material is washed into the next section. The material left in the final section is passed through a small clean-up box placed in the main one. The lower portion of the sluice box is only cleaned up once a season. Cleaning up takes place during the day time; the riffles are often replaced during the night. If there is not time to complete a clean-up by the end of the day, gravel is sluiced on to the untouched portion to discourage unauthorized persons from assisting in the clean-up.

Hydraulic Dredging. When alluvial is just below the level of the surface drainage, a modification of ordinary hydraulic sluicing is used, suitable for small isolated deposits, or for re-working old diggings; when finished with the ground is nicely levelled. The pressure of water forced through the nozzles may be natural or more often induced by a centrifugal pump. The plant is simple and inexpensive. As the ground to be worked is liable to be flooded, and the plant has to be moved about as work advances, it is mounted on a barge about 50 by 40 ft which, when loaded, draws 3 ft of water.

Work is commenced by making an excavation a little larger than the barge and 6 ft deep which is then filled with water. The barge is built on one side, and when finished is launched into the hole, after which the machinery is placed on board and the whole housed in. The machinery consists of a nozzle pump and a gravel pump, each with its own vertical engine, that for the nozzle being 175 b.h.p. having 450 r.p.m., while the gravel pump is 130 b.h.p. with 435 r.p.m. If driven by steam there are two multi-tubular boilers, 14 by 5½ ft designed for a daily working pressure of 160 lb/sq in. above atmospheric pressure. There are also a washing down pump, a feed water pump, a bilge pump, feed water heater, condenser, and air pump. The nozzle pump is a 12 in. multi-stage pump which delivers water at a pressure of 60 to 70 lb/sq in. at the face through a nozzle 4½ to 5 in. diameter. The centrifugal pump for elevating gravel is a single-stage type, the impeller of which is made wider than for lifting water only; the space between the impeller blades and the casing is also greater. Gravel pumps are known by the size of the suction pipe, 6, 8, 10, or 12 in. diameter. These pumps may have a renewable manganese-steel liner which protects the casing from wear, and the impeller can be fitted with renewable blades. The impeller may or may not be shrouded. As manganese-steel is too hard to be machined, it has to be cast. Sometimes centrifugal pumps are lined with rubber. The wear and tear of the gravel pump is naturally very great:

it is capable of passing boulders up to 50 lb. The bearings are made sand-proof by means of clean water conveyed to them under greater pressure than that at which the pump is working.

By the time the machinery has been placed on the barge, another excavation has been made alongside the first down to bed rock, and a level bed of logs made for the barge to rest on. This is then filled with water and the barge floated in, the water is pumped out, and the barge settles into position. A sump is sunk about 6 to 8 ft deep, the depth depending largely on the hardness of the rock, and a small cask is thrown into the water to prevent air from being drawn into the suction pipe. The sluice from the face to the sump, and also its branches, are given a grade of 1 in 24. An iron grating near the sump prevents stray boulders too large for the gravel pump, from passing into the sump. It takes two to four weeks to make the first cut. There is a crane at the end of the barge for lifting heavy weights. Most plants have an electric light installation, the dynamo being wound for 35 amp and 65 volts. Incandescent lamps are used on the barge and arc-lights at the workings which is driven by a separate engine. The sluice box has a total length of 80 to 100 ft, it is 4 to 4½ ft wide, and 13 in. deep. The sluice box is made of sheet-steel ⅛ to $\frac{1}{16}$ in. thick and made in lengths of 10 to 12 ft, so that they can be readily taken apart and re-erected when the barge is moved forward. At first the tailings are stacked on the surface, but later they are run into worked out places behind a brush and sand dam erected at the back of the barge. Water is drained off from the tailings through a large wrought-iron pipe which is eventually buried; and this enables about two-thirds of the water required to be used over again. Under ordinary conditions the ground is worked in paddocks of one or two acres, after which water is led into the paddock sufficient to float the barge to a new site, and the sluice box is moved farther up. Timber is often met with while hydraulicing, but it is a mistake to use it for fuel, as pyrites are generally deposited on it, and this soon causes the destruction of the boilers. If the height from the sump to the sluice box is over 80 ft, it is necessary to use two gravel pumps in series, the lower delivering direct to the upper. When running at 350 r.p.m. a 10 in. gravel pump will lift 1 cu ft of gravel and 400 cu ft water per min. The minimum quantity of water that can be used per cu yd of gravel is about 2000 gal. Instead of a gravel pump, an hydraulic elevator can be used. This is an injector consisting of an inclined pipe, into the bottom of which a jet of water under high pressure is allowed to play. In addition to the force due to the head of water, a suction is created which draws up the gravel and water collected in the sump. The head of water may be natural or induced by a centrifugal pump. The effective work of an elevator is from 10 to 15 ft for every 100 ft of hydraulic pressure, and 1 cu ft of ordinary gravel requires 2 cu ft water issuing from the jet to lift a maximum height of 15 ft. The diameter of the nozzle of the injector varies from 1½ to 3¼ in. A strong removable cast-iron liner is inserted in the lower part of the uptake called the throat, and is made to contract so as to prevent the scattering of the jet, after which it gradually widens out again. It is in this throat that the greatest amount of friction takes place. A liner 3 in. thick lasts about six weeks. The Evans' elevator has an auxiliary suction on each side of the main suction pipe as they tend to maintain a uniform velocity of water and gravel. It is necessary to have a certain amount of air in the water to work an injector satisfactorily, but it has been found impossible to regulate this amount through the mouth of the elevator. To rectify this, W. R. Crandall has added what he terms an atmospheric jet which surrounds the top of the water nozzle.

Flow of Water in a Ditch or Race

$$Q = ac\sqrt{ra}$$

Q = quantity of water which the ditch or race is capable of carrying in cubic feet per second
a = the effective area of cross-section of ditch as originally constructed, in square feet
r = the hydraulic mean depth in feet
s = the fall of surface in a unit of length
c = a coefficient covering all common losses (absorption, percolation, evaporation, etc.) which reduce the flow.

Loss Due to Seepage

This depends on the nature of the ground. If the ditch is below ground-water level, there will be an inflow from the adjoining land, and vice versa if the ditch is above water level. If the ditch is above water level, the approximate leakage may be obtained from Etcheverry's table of loss in 24 hours per square foot wetted perimeter.

Impervious clay loam	0·25%–0·35%
Ordinary clay loam, silt, or lava ash loam	0·50%–0·75%
Gravelly clay, loam, or cemented gravel	0·75%–1·00%
Sandy loam	1·00%–1·50%
Loose and sandy soils	1·50%–1·75%
Gravelly sandy soils	2·00%–2·50%
Porous gravelly soils	2·50%–3·00%
Very gravelly soils	3·00%–6·00%

Bucket Dredging. This method is used in places where there is too much water in or over the alluvial for the employment of some form of hydraulic sluicing; or if there is not sufficient water or pressure for sluicing, such as in river flats or rivers. A dredge that has served its purpose under certain conditions may be quite unsuitable under other conditions. Before deciding on the design of a dredge, one should ascertain the value and distribution of the gold or tin, the quantity, the depth, area, nature of the ground to be worked, the character and contour of the bed rock. When the deposit is covered with water, one should find out the usual depth, the liability to floods, whether there are many boulders or snags, the timber growth on the land: also data common to all mines, such as probable operating costs, class of labour, supplies, transport, cost of land, royalties, etc. One must distinguish between tight ground and cemented ground. The former may be loosened by explosives; the latter is not a dredging proposition. The bed rock should be soft for dredging purposes, and even then it is doubtful if the bottom is cleaned up as well as by hydraulicing. Clay beds are objectionable, not only because the clay is difficult to get out of the buckets, but because the clay, if not readily dissolved, forms balls which carry away grains of gold. Boulders and snags cause delays and breakages.

If rock bars occur in a river, they may have to be blasted, or a dam built to back up sufficient water to allow the dredge to float over the bar; otherwise the dredge remains idle till a flood comes. Freshets may wreck the dredge unless it can get under shelter in time; or the hole in which it is working may become silted up. Bucket dredging does not add material to rivers or flats as does hydraulic sluicing, but in turning it over the loosened material is increased in bulk. If the river is deep, the tailings can be deposited behind the dredge: but if working a paddock, a stacker is required and the worked ground can be resoiled as the dredge goes forward.

The hull, formerly made of wood, is now generally made of steel. It consists of two long pontoons, with a shorter one between them. This leaves a gap between the two long pontoons in which the bucket ladder can operate. If the dredge is to work in a river, the bows are made square, since experience proves that it is steadier than when the bows are rounded. If the dredge is to work in a paddock—as most dredges do—since the dredge has to cut its own flotation, the bows are rounded to enable it to work into corners. The bottom of the bows slant upward to help any snags or boulders brought up to be easily handled. The bows should be protected by heavy planks or steel plates against timber floating down in time of flood, and in the case of a paddock dredge to preserve it from damage by striking the face as the dredge surges about. The buoyancy of the hull must be calculated, and the weight it has to carry be so distributed that when the ladder is down and at work, the deck will be level, otherwise water on the concentrating tables will not be evenly distributed. When the bucket ladder is up, the centre of gravity is naturally altered; the bows will be pressed down, but that will not matter, as the tables will not be working then. The level of the tables may be adjusted by the stacking of fuel on dredges when steam is the motive power; or by shifting boulders about the deck. The hull should be made as stiff as possible in order to withstand the strains to which it is subjected. The port and starboard pontoons should each be divided into three water-tight compartments so as to minimize the chance of sinking; for when once a dredge has been under water, not only is there the expense of raising and repairing it, but the time lost has to be taken into consideration, and the machinery never runs so well after such an accident as before.

The hull, however, is provided with sea-cocks so that it can be purposely sunk in case of fire. The length of the hull largely depends on the dredging depth. One must distinguish between *dredging depth* and *depth of gravel*, as they are often confused. The dredging depth is the depth between the water line where the bucket ladder cuts it, and the cutting edge of the bottom bucket when the ladder is lying at an angle of 45°. The width of beam is generally 50 to 58 ft with an overhanging deck of 5 or 6 ft. It draws from 3 to 5½ ft of water. Besides connecting the three pontoons with floor and deck beams, there are other frames which form thwartship girders. There is the bow or forward gantry for raising and lowering the bucket ladder: it may be A-shaped or consist of four parallel uprights connected on the top and properly braced.

The A-shaped pattern is not so good because it tends to spread the pontoons apart. The tumbler frame is situated more or less in the middle of the hull. The distance of the tumbler frame from the bows depends on the length of the ladder, and by noting this from a distance, one can form an idea of the dredging depth. The top tumbler is generally 25 ft above the deck. If it is necessary to lower the centre of gravity and the tumbler is placed lower down, the screenings will have to be elevated to the tables. Sometimes cranes are erected for handling logs. On those dredges where stackers are used, a stern or after-gantry is built to support it. Dredges have a tendency to *hog*, i.e. the stem and stern are inclined to drop. To counteract this, a hog framing is built, generally inside and higher than the tumbler frame. Steel rods or steel ropes tightened by tension screws lead from this fore and aft. Extra hog frames are used if the hull is divided very far up by the ladder well.

The bucket or digging ladder is constructed of steel and varies in length according to the depth of the ground and water overlying it. The ladder is provided with heavy rollers spaced from 5 to 7 ft apart so as to reduce the friction of the buckets. The top end of the ladder is suspended on a pivot, while the lower end is free to move up and down as desired. There are many designs for ladders. Care must be taken to make them strong and overcome the tendency to twist. In places where the water is deep at certain seasons, and comparatively shallow at other times, an adjustable ladder may be constructed from which a section can be removed, also of the bucket chain, and the shortened ladder worked from another gantry nearer the top tumbler. The top tumbler is of solid cast steel and is made four-sided, since it is found that the bucket chain clings better to that shape than if it more nearly approached a cylinder. This is important should the bucket chain snap for, if the winchman acts quickly he can prevent the whole chain from running off. The bottom tumbler at the lower end of the ladder acts as a roller for the buckets to pass round. It has five sides as this form presents the buckets at the best angle for filling. The bottom tumbler should have large flanges to prevent the bucket chain from working off. The bottom tumbler having to work in water thick with gravel and sand wears out much more quickly than the upper tumbler. A paddock dredge has a projecting ladder so that the buckets can eat into the bank: if this were not so, it would be impossible for the dredge to cut her own flotation and work into corners.

Buckets for heavy work are made of cast manganese-steel. For lighter work they are built up of bottom, back-hood, and lip. The bottom is made of high carbon- or nickel-steel: the hood may be a solid casting or forged piece; the lip, which has the hardest wear and most frequently has to be renewed, is made of manganese-steel. Buckets vary in shape, comparative length, depth, and angle of lip. The main advantages of a large bucket are increased yardage (which is necessary in swiftly flowing streams) with practically the same labour, and the possibility of handling bigger boulders. The disadvantages are greater first cost, the necessity for having special facilities for handling heavy dredge parts, increased difficulty in washing the larger and highly irregular amount of material delivered at a time to the screen. Those buckets which are narrow and deep do not empty so well as those which are wider and shallower, especially if the material is clay. With clayey material, a heavy hinged false bottom is sometimes used to assist in pushing the material out when the bucket is reversed. Jets of water are sometimes arranged above where the buckets empty to help in cleaning them out. The buckets may be close or open connected. The open connected have a link between each bucket and empty at the rate of 10 to 14 per min. The bottom of the close connected buckets form the link, and they dump at the rate of 18 to 20 per min. There is more vibration in dredging with open buckets than with close connected buckets. It takes 14 per cent more power to run a close connected chain than an open connected. It has been found that the best results are obtained with a chain speed of 30 ft/min in tight, and 35 ft/min in loose gravel. When a greater speed is attempted, it is difficult to fill the buckets and the actual capacity is less. Open connected buckets fill better than close connected, but there is only half the number of buckets, so even when run faster, the dredging capacity is less. A grab hook can be inserted instead of a bucket in open connected buckets which is useful when snags are encountered. A friction adjustment is used in connection with the upper tumbler so that when any undue strain comes on the gearing it slips and stops working until the strain on the bucket chain is relieved. The size of the buckets depends on the depth to be dredged, the nature of the wash, whether the bottom is even or uneven, and whether the dredge has to work in a strong current or not. Modern dredgers are not in favour of very large buckets. For an average depth of 35 ft it would probably be uneconomical to have buckets larger than 9 cu ft: but for ground 65 ft and over, a 16 cu ft capacity might be advisable.

Tin dredges are generally larger than those used for gold; they also require a larger crew. When speaking of a yard of ground, unless otherwise stated, a yard in situ is meant, for when loose it occupies more space. Dredging is started at the lower end of a lease so that it shall not be necessary to rehandle tailings.

The dredgmaster must use judgment in deciding on the quantity of material it is advisable to lift at a time. In the case of very rich or clayey material, the gold has a better chance of being saved if treated in small quantities: but if the wash is poor, it is better to work the dredge to its full capacity. The bucket chain is allowed to drag on the ground so that about three buckets are in contact with the ground at once. If a snag gets in the way, the gravel is removed from about it as much as possible, and the snag brought to the surface by grab hooks, when it is sawed into lengths and removed to the rear of the dredge. Should a boulder be encountered which is too large to be raised in a bucket, one must work round it and allow it to fall behind.

Where there are high banks (15 to 40 ft high), the face above the water is broken down with a jet of water from a nozzle, the pressure being obtained artificially. If the wash is so stiff that the face is practically vertical, the buckets must undercut the face to make it fall. This requires much skill to avoid an overhanging face from burying the bucket ladder. Should such an accident happen, the ladder should be raised as quickly as possible before the material settles solidly around it. Should the bottom be too deep in places for a paddock bucket ladder to reach it, it may be possible to pump out some of the water to lower the hull sufficiently to reach the bottom.

The most common accident on a dredge is a broken pin; if two pins connecting a bucket break, the chain may fall to the bottom and get tangled up. Another accident is when the bucket chain gets off the lower tumbler owing to the chain being too loose.

A dredge is handled with six steel wire ropes. Looking towards the bows, those on the left-hand side are known as the port forward or bow line, and the port after or stern or quarter line: those on the right-hand side are called the starboard bow line, and the starboard stern or quarter line: the head line in front of the dredge, and the ladder line. Each is attached to its special drum, the free end being fastened to trees or rocks if suitably situated, otherwise to "dead men." The head line is anchored 900 to 1200 ft ahead of the dredge so as to give a good sweep. If too short it makes a curved face. The wider the face the better, since taking out corners occupies much more time than straight dredging. The head line, which is stronger than the side lines, is drawn up 2 or 3 ft after each cut across the face. Head lines are all very well for soft ground, but when the ground is tough, hard, or cemented, spuds are preferred as they hold the dredge more steadily against the face and obviate the surging of the dredge as the buckets strike and leave the face which tends to sink the stern and push the end of the sluice under water.

Spuds may be of wood or structural steel: they have a pointed shoe at the lower end and sheaves at the upper end so that they can be raised and lowered by ropes. In section they are generally 24 by 30 in.; they work in guides at the stern of the dredge. Two spuds are used to a dredge; one is known as the digging spud and is generally made of steel and placed on the starboard side; the other, known as the stepping spud, often made of Oregon pine placed on the port side of the stern, or sometimes on the centre line. The spud steps ahead about 10 ft when the ground is soft, and 7 ft when hard. When working with spuds, the face is in the form of an arc with about 100 ft radius. It costs more to equip a dredge with spuds than with a head line, but little time is lost moving ahead with spuds, the labour of burying dead men for the head line is avoided, and spuds hold the dredge more steadily against the face. The port and starboard lines are generally 1 in. diameter, and the head line 1½ in. The side lines are more flexible than the head line, and on account of the ladder line having to pass over so many sheaves, it is more flexible than the side lines.

The material dredged passes into a revolving screen to wash and separate the gravel from the sand. These trommels are 15 to 25 ft long and 5 ft in diameter, having perforations $\frac{5}{16}$ to $\frac{3}{16}$ in. diameter at the upper end, and $\frac{3}{8}$ to $\frac{5}{8}$ in. at the lower end. It is usual to have a single trommel arranged lengthways in the centre of the hull at the head of the tables. A sparge pipe inside the trommel supplies the necessary water for the tables. About 250 cu ft of water is required to wash 1 cu yd of washdirt. The trommels are given a pitch of 1 in 15 and revolve on 12 in. diam. rollers. Sometimes instead of a trommel two shaking screens are used. The concentrating plant is generally placed on the dredge, but sometimes it is placed on a scow: in the latter case the tables can be kept level all the time. When the tables are on the dredge they are arranged on one side only. The tables are generally made of sheet iron divided into 3 ft lengths by partitions 3 in. high to prevent water from forming channels. They are given a fall of 1½ in 12. The bottom of the table is covered with canvas or coarse calico, and over this is placed well-made coconut matting with the cross pattern upward. This matting is cut into lengths of 3 to 4 ft for convenience in washing. Expanded metal is placed on the top of the coconut matting; it is laid with the length of the holes at right angles to the length of the table. The direction in which the least resistance is felt when rubbing the hand over it is the direction in which the material should flow over it. Most of the gold is caught in the first two divisions. Many other kinds of riffles are also used.

A plant suitable for gold saving is not suitable for saving tin; because the concentrates of tin are so much more bulky than those of gold that it would occupy too much time cleaning up. Tin is generally saved in a sluice box about 120 ft long, 6 ft wide and 2 to 2½ ft high. About 40 ft of the box are supported on the barge, the rest being on trestles on the ground. The best fall for saving tin is 5 in. in 12 ft. Stops about 3 in. high are placed in the sluice box some 16 ft apart which fit in slots made of angle iron riveted to the sides of the sluice box. Men stir up the gravel and when tin begins to show, another set of stops is placed in position, and the stirring up repeated. This is known as *reducing*. When the sluice box is full, the men commence *running down*. This is similar to reducing, only instead of working from below upward, the process is reversed, so that any tin which escapes from the upper divisions has a chance to be caught lower down. The tin is generally concentrated to within 30 and 50 per cent in the sluice box. It does not pay to make it cleaner in the sluice box as it would take too much time from the working of the dredge. The concentrates are further cleaned in a streaming down box, which is simply a small sluice box, and finally made ready for the market in a Willoughby. In some plants they use tables with mechanical rakes, classifiers, and a modification of the Hartz jig.

Quarrying is generally used for obtaining building stone, road metal, clay for brick-making, limestone for the manufacture of Portland cement, etc. When used for mining ore on the surface, or for obtaining filling material, the excavation is generally called an open-cut. The method of quarrying depends largely on the thickness of the deposit; the material to be mined; the purpose for which the material is wanted; the output required; and whether the deposit is on a hilly or flat surface. If the rock is to be used for building purposes, advantage is taken of joints and bedding planes which assist in breaking the rock with the least possible strain.

With some rocks such as slate, there is a considerable amount of wastage; while with gravel pits and rock mined for road metal the loss is not so great. Soft dimension rocks such as marble can be cut up into blocks with a wire saw, and divided up again by the same method into marketable sizes. This consists of an endless wire composed of three helical strands of high tensile steel, from $\frac{5}{32}$ to $\frac{1}{32}$ in. diameter, which travels in a groove at the rate of 1250 to 1550 ft/min. The wire may be of any length up to a mile or even more according to circumstances. Pulleys round which the wire circulates are mounted on pillars which are lowered into pits cut out with chilled steel shot. These pulleys are $2\frac{1}{2}$ ft diam., one at the top of the pillar, the other capable of being moved down as the cutting proceeds. The cutting is actually done by wet sand, the wire acting as a conveyer. The sand should all pass through a 35 screen and 90 per cent be left on a 100 screen. Sixteen to twenty pounds of sand is required per square foot cut. If the cut is more than 50 ft in length the abrasive should be fed at several points. A good wire 1090 yd long will cut 1600 sq ft before requiring replacement. Hard rocks may be divided by a series of plugs and feathers. Holes from 3 to 5 in. deep are drilled 6 to 8 in. apart, in each of which is a plug with its feathers which is hammered progressively and splits the rock. Or better still, instead of drilling a hole, use an air wedge-hammer which gives a lead to the wedges when driven in. In cold climates holes are filled with water and allowed to freeze during the night. Chanellers are also used for obtaining blocks of dimension stone.

One must consider the ultimate boundary of the excavation, otherwise buildings may be erected within it, the extra cost of removing which may stop working by open cutting which otherwise would be the cheapest method of mining. The dip of the deposit, if on the side of a hill, affects the cost of mining; for if dipping into the side of a hill, there is a greater amount of overburden to be removed than if dipping away from the hill. Quarrying may be carried out by benching or tunnelling. Quarrying by benches is employed where the deposit is of considerable thickness. The height and width of a bench will depend largely on the strength of the material to be quarried, and the width must be sufficient to prevent the material brought down running over to the bench below. A deposit of, say, 50 ft thickness can be mined in one bench. It is seldom possible to carry a vertical face, so it must be given a batter. If the area is large and the deposit thick, it may be worked in the form of a spiral which serves as a track for a steam or electrically driven train. Bench mining is not very suitable for rocks having highly inclined strata.

If the overburden is shallow, and the deposit is not very deep, the soil may be ploughed in strips and removed by scoops, drays, trucks, or other available means. When the ore is uncovered it can be mined and raised by a travelling crane, steam shovel, or flying fox, or drawn up an incline in cars. When the ore is worked out of one strip, soil from the next strip is dumped into the excavation and the ore of the second strip mined. This is repeated till all the ore of the area is won.

There are two methods of blasting down a face of rock or ore: one by a series of deep bore holes at the back of the face, the other by tunnel blasting. The bore holes, which are about 6 in. diam., should be deep enough to loosen the material right down to the floor of the bench, for if this is not done, the bottom will have to be shot up later. The distance apart of the holes, and the distance back from the face, depend on the nature of the rock. At first one may try spacing holes from the face and from each other three-quarters of the depth which may be satisfactory for sedimentary rocks of medium hardness, but not for granite. Broken rock should lie just in front and below the position of the holes: if scattered it shows too much explosive has been used. There are always large blocks which require further reduction in size. After blasting, the face is barred down before shovelling begins. This is done by men connected by ropes to some anchorage at the top, who commence at the top and work down. As each bore hole is finished, it is covered by a slab of rock and cemented all round the edges to prevent anything from falling in. It is found that column loading gives a better result than springing or bulling the holes. Giant blasting should not be used for cuts less than 60 ft. The principle of tunnel blasting is to blow out the toe and cause the superstructure to fall. A tunnel is driven into the face, level with the bench for a distance not greater than the height of the face, and is given a grade of 1 in 50 for the sake of drainage. At the end of this tunnel there is a drift on either side parallel to the face. The drifts are made as small as possible (4 ft high and 2½ ft wide). These are charged with explosives and tamped. Two fuses are laid in case one fails. One pound of black powder will bring down 5 tons of limestone, while 1 lb of high explosive will break about 7 tons of a similar class of rock; but more explosive is required later on to break up the blocks which are too large to handle. This method is not suitable for horizontally bedded formations; there is danger when the toe is blown out leaving the overhanging bedded formation to fall by its own weight instead of being blown forward.

Milling can be used in either hilly or level country. It differs from benching inasmuch as, although the material is broken at the surface, it is conveyed underground for transport by either an adit or a shaft. A rise about 6 by 4 ft is made from a drift to the surface where the ore is to be mined. If the material is likely to pack it is better to make the rise inclined instead of vertical. About 10 to 15 ft from the surface logs are placed across the rise to serve as a grizzly. The material is then broken down in the form of an inverted cone known as a glory hole. The finer material passes through the grizzly, the larger pieces have to be broken up sufficiently to fall through. When the mined material will no longer roll down the side of the cone, the grizzly is lowered 10 ft and the cone is enlarged. This process is continued as far down as possible. To work the cone of solid material left between glory holes, a rise is put up in the middle and the block worked on the same principle as before. When this secondary cone is worked out, there will still be some solid material left. This can be mined and loaded into cars from a *chinaman* instead of through a pass. Care must be taken that the partition between adjoining cones is sufficiently high to prevent pieces rolling down in one from running up the opposite side and over into the adjoining cone, where it might hurt workers. The passes should never be allowed to stand full, but must be kept on the run to prevent packing.

The advantages of milling are: the large tonnage per man, the minimum amount of handling, and the low cost. The disadvantages are that the passes are liable to block, and that rain water passes underground. The advantages of open cuts generally are: the equipment and development costs are light; the production stage is reached quickly; the minimum of skilled labour is required; supervision easy; no trouble about ventilation; no loss of material in pillars; work can be stopped and started at any time; all holes are down holes; no artificial lights necessary unless worked at night. Its disadvantages are: when the surface is valuable, much damage is done, work has to be suspended during unsuitable weather, and if the workings are below surface drainage level, they are liable to become flooded.

A time comes in most cases when a calculation has to be made whether it will be cheaper to continue open cut mining, or to start some underground method. The sides of an open cut must not have one steep slope, but should be stepped back every 30 to 50 ft in height, leaving a bench sufficiently wide to stop any pieces that break away from falling to the bottom, for even a small stone falling from a height can give a dangerous blow. As a batter has to be left in most open cuts, the mouth of the cut extends with depth, and sooner or later the cost of removing the overburden becomes greater than the value of the ore exposed; for the ore won has to be debited with its proportion of the overburden.

Lay-out of a Mine

The lay-out of a mine has an important effect on its future so it should not be left in the hands of an inexperienced man. Even if a capable foreman is placed in charge of the preliminary work, he ought to be under the control of a mining engineer who should make frequent inspections. A badly located shaft may be the cause of a constant expense for transport underground. An adit or shaft should be of suitable size. If made too small, the output is restricted which may mean that another passage will have to be constructed. If too large, the cost is increased unnecessarily. Factors to be taken into consideration are the nature of the mineral body, its width, whether it nearly approaches the vertical or horizontal, and whether its walls will stand well; surface features, whether hilly or flat, near bodies of water, or places where the surface must not be disturbed: the timber and labour available: whether the demand for the product of the mine is likely to be continuous or intermittent.

The best method of approach to the deposit must be carefully considered, whether by open cut, adit, inclined or vertical shaft. The first cost, upkeep, and working expenses of different methods must be worked out, not guessed. The object of mining is to win minerals at a profit. A mine is not the substance, but the mode of obtaining the substance: therefore it is wrong to speak of owning a mine, when it is only an unworked claim. Mining is the art of extracting valuable minerals from the Earth's crust, no matter whether the operations are carried out on the surface by open cut, hydraulicing, dredging, or by some underground method.

Excavations, such as cuttings of tunnels for railway purposes, are not considered mining from a technical point of view, though the principles used are the same.

The entrance to a mine should be readily accessible, so that stores, machinery, fuel, etc., can be brought there, and products of the mine taken away. There should be ample room about the mouth of an adit or shaft so as not to be cramped, and cuttings or made ground be required. The mouth of an adit should be sufficiently high above a creek to prevent flood water entering it. There should be sufficient room for ore-bins and mullock heaps, and the latter should not be dumped on the outcrop of a lode which might otherwise be worth working later on. When there is no natural fall on the surface, the ore or mullock may be raised above the level of the ground by hoisting to a brace. When the entrance to a mine is through an adit, there is generally a natural tip, and this can be increased if necessary by giving the track an inclination of 5 per cent. The entrance to a mine, besides being above flood level when possible, should be away from any possible landslip or avalanche. If any part of the workings is likely to be under swamps or a body of water, as in the case of some coal mines, the necessary cover of rock between the water and deposit must be considered, and allowance made for possible depressions.

An underground mine is divided into blocks by horizontal and vertical passages. The former are known as drifts (sometimes erroneously called levels), entries (in coal mines), and cross-cuts: the latter as shafts, and winzes (rises). The better a mine is opened up, the larger the number of points from which ore can be obtained, and as in many cases the ore is not uniform, this enables the different grades of ore to be blended suitably for subsequent treatment. Though many mines are developed two or three years ahead to assure a regular supply of ore, it is a mistake to over-develop a mine as the cost of upkeep added to the interest on the cost of the outlay may counterbalance any advantage derived from the development.

The different levels take a certain shaft as their datum, and obtain their names in numerical order; or better still, according to their depth. From these levels, drifts are driven, either in the ore itself, or in the footwall rock, approximately parallel to and a few feet away from the lode. Rock drifts, being straighter than those which follow the curvature of the lode, are better for trucking and ventilation and cost less for upkeep. Where there is a group of mines, they sometimes combine to drive an adit so as to drain their section of the country. In rough country, it may be cheaper to drive a long adit and start a blind shaft at the end than to sink a shaft from a greater height; for though it is quicker to hoist than to haul, yet several cars can follow one after the other or be made into trains and be propelled by man, horse or mechanical means. The dimensions of a drift or entry depend on the expected output, the motive power, whether single or double tracks are required, and the strength of the rock driven through. If the rock is weak, as in some collieries, it may be necessary to have smaller but more entries, not only for traffic purposes but also to obtain sufficient air without the current being too strong.

If water is likely to be encountered in a drift, it is cheaper to cut a ditch on one side while driving than to come back later on to do so. If the quantity of water to be drained is large, as in deep leads, it may be necessary to have a drain below the foot-boards; but this is not advisable if it can be avoided. In some cases a drift may have to be divided by a brattice, either at the side or overhead. In the latter case the drift will have to be made higher than usual; or room may have to be left for an air pipe. Single track drifts should be at least 6 to 6½ ft high, 3½ ft wide at the top and 4½ to 5 ft at the bottom. Double tracks are usually 7½ ft high by 8 to 10 ft wide within timbers. Double tracks are used in mines with a large output, such as in collieries, unless the drifts are very long, when there may be a single track with passing places. Double tracks are also laid down where traffic is likely to be congested, such as in the main cross-cut which receives cars from the drift on either side of it, and where empty cars can be kept till wanted. In double track drifts, there should be sufficient clearance for a centre prop if required. Refuge holes should be cut out of a side of an entry, whitewashed so as to be readily seen, and kept clear of timber and rubbish. If cars are drawn by animals, these refuge holes should not be more than 150 ft apart: with mechanical haulage they should be spaced at most 60 ft apart.

The cost per lineal foot is somewhat greater for driving a double track passage than a single one, but not double the cost, as there is the same number of tight corners in both cases. The cost per cubic yard works out less for a double than for a single track. Drifts should have a slight upward grade as they proceed inward, partly to make it easier for loaded trucks to be handled, and partly to assist drainage: but it must not be so great that cars are liable to get out of control. Moreover, one has to consider cars going inward loaded with timber, drill-steel, machines, etc. The usual grade for hand tramming is 1 per cent, and for mechanical hauling ½ per cent. The vertical distance depends on one or more of the following: the size of the deposit: its underlie: the way values occur: the hardness of the rock. In the early life of a mine, the levels are often placed 50 to 75 ft apart, because the owners wish to obtain ore as soon as possible to help pay expenses. The usual distance apart is 100 ft, but as one goes deeper, it may be advisable to increase the distance to 200 or 300 ft to save development costs as the cross-cuts become longer from the shaft: or some change in the method of mining may make it desirable. But there must be a limit; for if too far apart, other expenses creep in; the men take too long in reaching their work; the ore passes are liable to become worn out before they have served their purpose; and the number of working faces are reduced.

When working a narrow vein with a flat underlie, say 35° from the horizontal, or less, the levels should not be more than 50 ft apart on the underlie, otherwise there will be a difficulty in passing ore to the level below. When the ore occurs in irregular bodies, the levels must be placed closer together. To save the cost of driving and upkeep as a lode recedes from a shaft, levels may be placed farther apart, and later an intermediate drift can be driven from a winze. Intermediate drifts are useful when the ground is bad standing and the ore has to be extracted quickly: also when mining small irregular deposits; and they may be necessary if a lode becomes faulted. In a wet mine, a drift should be driven below where the ore is being extracted so as to give the stopes a chance to drain.

Cross-cuts should be as short as possible, and are more or less at right angles to a lode. As they generally cross the grain of the country, they require less support than ordinary drifts, and the roof is arched. The object of the cross-cut may be to connect the shaft with the lode, in which case it is generally made wider than other cross-cuts; to determine the width of a lode; to divide a lode up into convenient blocks for ore extraction; for exploratory purposes; to connect a rock drift with a lode. In many cases it is advisable to make a curve where a cross-cut meets a drift, instead of using a flat sheet.

Shafts may be vertical or inclined. Sometimes a shaft is started vertically, but on reaching the lode it continues on the underlie. It is then known as a "broken shaft." If a vertical shaft is not continued to the bottom of the mine, but the lower section is offset, the lower section is called a "counter shaft." A shaft may be used for a special purpose, such as for hoisting ore, lowering and raising men, ventilation, or pumping; or one shaft may be used for two or more of these purposes. There should be a double exit from every mine in case of an accident; also for ventilation purposes. If two shafts are to be had the question is whether all the hoisting shall be done through one shaft or shared with the other. When possible, men should travel through the down-cast shaft for the sake of fresh air. In general practice, it is more economical to concentrate the traffic at one shaft. It is sometimes advisable to test the ground with a diamond drill before sinking a main shaft, as this gives a sample of the rocks to be passed through and indicates whether watery strata are likely to be encountered. The presence of known faults must be noted, as one does not want to sink in disturbed country if it can be avoided. Where possible, vertical shafts should be so situated as not to pass through the lode; but where this cannot be avoided, try to arrange that it passes through a poor part so as not to lock up rich ore in the shaft pillar. Vertical shafts are generally started on the hanging-wall side of a steeply inclined narrow lode. The distance of the lode from the shaft on the surface depends largely on the depth it is desired to penetrate the lode, and this is subject to the length of crosscuts which are generally arranged to be longer in the upper levels than deeper down where the rocks are likely to be denser. A shaft pillar of sufficient strength must be left round a shaft where it passes through a lode, and this should not be robbed or weakened till the shaft is to be abandoned. In a narrow vein, this pillar may be from 12 to 30 ft or more on both sides of the shaft. Various formulae have been published for determining the size of shaft pillars, but none is reliable, and at best they can only be approximate as one cannot place correct values on the various joints in the different beds, and may overlook faults and other planes of weakness.

In coming to conclusions, the mining engineer is influenced by the width of the lode, the strength of the material composing it, the depth from the surface, the underlie of the deposit, and the nature of the rock above and below. The angle of fracture of a rock varies with its nature, being steeper in hard rocks like granite and sandstone than in softer rocks such as shale. The thicker the deposit, the larger the area of the pillar required for supporting equal weights, since a high pillar is more likely to fracture than a short one. There should be more of the base of a pillar to the rise than to the dip. Perhaps Bron's rule for shaft pillars in coal mines is as good as any to serve as a basis for coal seams and ore beds, to be modified according to circumstances. He draws a line to enclose all surface buildings about the head of a shaft, and drops perpendiculars to the bottom of the deposit. He then increases the size of the pillar thus formed by one-third of the depth of the shaft all round.

The tendency in modern times is to extract all the ore round the shaft of a metal mine during sinking, and to fill the space with concrete. If the lode to be worked is very wide, and the excavated ground is likely to cave, it is better to sink in the foot-wall, instead of the hanging-wall, side. If a shaft is required to serve more than one lode, the conditions become more complex, since one has to be careful that a loose block is not formed due to the mining. If the deposit to be worked is a bed, the ideal position would be about the centre of the lease, so that equal areas could be worked all round the shaft: but circumstances may make it advisable to sink in the deepest part of the deposit for the sake of drainage, or there may be some surface advantage in sinking elsewhere. In the case of collieries, the downcast and upcast shafts are placed fairly close together for the sake of ventilation. In vein mining shafts are also placed close together: but for deep mining, main shafts are placed farther apart to save expense. In deep alluvial mining, shafts are sunk in the rim rock on the side where the rock is steepest, as the shaft is then nearer the wash. The shafts in such deposits may be a mile apart, the distance to a large extent being governed by the grade of the deposit. The depth shafts sunk below the wash depends also on the grade of the old river channel, for it is not desirable that the rock drift should cut into the wash too soon.

The cross-section of a shaft may be rectangular, circular or elliptical, depending on the nature of the ground, whether watery or loose; whether ventilation is of great importance, and the material available for lining it. The area of a shaft depends on the quantity of ore to be hoisted in a given time, the quantity of water to be pumped, and the amount of air required for ventilation purposes. A rectangular shaft is lined with timber, and its area is better divided up than any other shape: but the air current is more or less interfered with by the cages or skips which occupy space in their respective compartments. Rectangular shafts generally have their longer axis parallel to the strike of the lode, partly because it is simpler for caging cars from the main cross-cut, and partly because it is easier to break the rock with the grain than against it. But if the rock does not stand well, the shaft may be sunk at right angles to the grain of the rock. Circular shafts will stand much greater pressure than rectangular shafts. They are lined with brick or concrete which offer less resistance to air currents than timber, and if water-bearing strata are encountered, cast-iron tubbing is used. In collieries, it is not usual to divide a circular shaft into closed compartments as that would interfere with the ventilation, but buntons are used to give support to cage guides, pipes and cables. In metal mines, a circular shaft, if divided into separate compartments, the space is not so well utilized as with rectangular shafts. However, those parts not used by cages, skips, and ladders, can be used for pipes, power cables, and ventilation. An elliptical shaft is a compromise between a rectangular and circular shaft. It is easier to divide up into compartments than a circular shaft. At the same time it can resist pressure better than a rectangular shaft.

It sometimes happens that a shaft has to be sunk at a point below flood level. In such a case, if there is no other connection with the surface, a concrete wall should be built round the shaft from a rock foundation to above flood level, but if this is not suitable, the mouth of the shaft is concreted, with a recess all round. In this recess are kept concrete covers which can reach across the shaft after the ropes, pipes and skids have been removed. When it is obvious that flood water will reach the mouth of the shaft, the covers are placed in position, made as watertight as possible, and held in place by props. There is, however, bound to be a leakage of some sort, and this would naturally compress air in the workings. To prevent this from blowing the covers off, a pipe is let into the concrete below the covers which is led up the headframe above flood level and serves as a safety valve.

An underlie shaft or slope should always be sunk when a deposit is 45° or less from the horizontal, but it is frequently used for steeper angles. An inclined shaft may commence at the surface or be a continuation of a vertical shaft, in which case it should be connected by a cubic parabolic curve (Fig. 34) so as to avoid having to slow down at the change of angle when hoisting, which would be necessary if a segment of a circle were used. An underlie shaft should be sunk in the foot-wall where it is not likely to be disturbed by the extraction of ore. It should be sunk at a uniform angle to assist regular winding. Cross-cuts from an underlie shaft to the lode are generally 20 to 50 ft long.

Winzes and rises merge into one another, and when completed are generally called winzes. They are more or less vertical passages connecting drifts. A winze may be sunk for various purposes, prospecting, ventilation, mullocking, to start stoping from, to ascertain whether it is worth while sinking a more expensive shaft, another lift, etc. When used as a pass to convey mullock from the surface to various levels, the mullock is diverted at each level to prevent it from packing deeper down. Winzes can generally be started sooner than a rise, as an upper level is naturally in advance of a lower. When required to hasten the work, winzing and rising go on at the same time if the lower drift is sufficiently far advanced. The starting points should be determined by a surveyor, for to measure a certain distance along two drifts without taking into account any change in the strike that may have taken place between them may lead to a crooked connection, known to miners as a "dog's hind leg" which is a frequent cause of hold-ups. Rising is quicker and cheaper than winzing, as there is no time wasted in hoisting broken material and water, and the ventilation is better. The break-through between winze and rise should be from the winze.

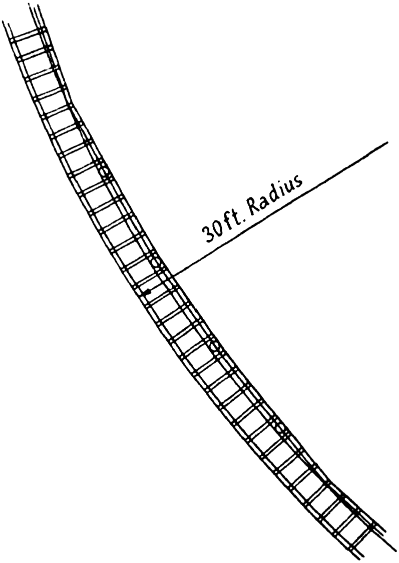


FIG. 34. CURVE JOINING VERTICAL AND UNDERLIE SHAFTS

Headframes

Headframes may be constructed of timber, steel, reinforced concrete, brick or stone, largely depending on what is available locally, and whether it is required to last for a few or many years. Timber is not uniform in strength, is liable to rot, and to be attacked by termites (so-called white ants) in certain districts if left unused for any length of time; besides, there is always a fire hazard. Steel is liable to corrosion: it should have a coat of paint before erection and two more coats after it is in position; it also requires repainting every two or three years. Steel lasts longer than timber, and being more uniform than timber it does not require such a high factor of safety; it has the further advantage of not being so liable to damage by fire. Steel headframes are mostly used for shafts that are expected to have a long life. Reinforced concrete does not add much to the strength, but does add to weight and stability; it is not uniform, so requires a higher factor of safety than steel. Being heavy, it requires a strong foundation. Brick or stone structures are walls, on the top of which is a small steel headframe for supporting the sheaves.

A headframe may be required for a vertical or inclined shaft. It may be pyramidal (Fig. 35 (*c*)), the four legs being given a spread in all directions: but if the spread is not sufficient, it may be necessary to add backstays. Such structures are lavish in the use of material, and are not suitable for steel. A gallows-frame (Fig. 35 (*a*)) consists of two or more legs (depending on the number of compartments) which are vertical and are supported by back-stays. Another set of legs is generally erected on the other side of the shaft which are tied to the main set and serve as a support for the sky-shaft which is a continuation of the shaft timbering above the surface (Fig. 35 (*b*)). There is not so much room on the brace or landing platform as in the case of a pyramidal headframe, so if more room is required an extra platform will have to be erected, which should be independent of the headframe so as not to be subject to its vibrations.

The height of a headframe depends on the height of the brace from the ground necessary for tipping purposes, the speed of hoisting, the height of the cage or skip, the length of bridle chains and safety hook, and the length of one revolution of the drum in case of over-winding. Headframes generally vary from 30 to 100 ft but are sometimes higher.

The maximum working or live load is the weight of the rope when at the bottom of the shaft, the full cage or skip, bridle chains, and safety hook, the impact on starting the hoist, the accelerating force, friction of the rope on its sheave and winding drum, and of shoes of cage or skip on the guides.

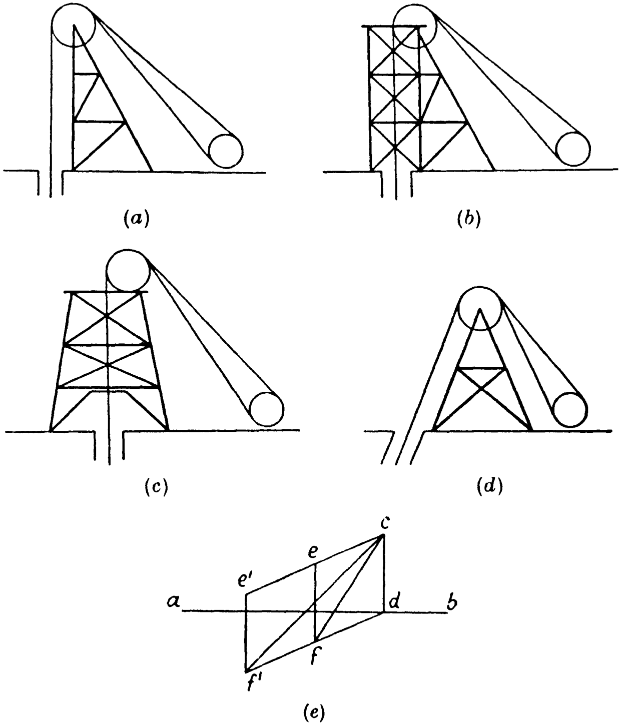


FIG. 35. HEADFRAMES

(*a*) Shear-leg; (*b*) Four post; (*c*) Pyramidal; (*d*) A-frame for inclined shaft; (*e*) Force diagram for headframe

The forces acting on the headframe are the pull of the rope under varying conditions, the weight of the structure, and the pressure of the wind. The maximum stress is equal to the breaking strength of the rope should the cage or skip become jammed in the shaft. Wind pressure under usual conditions is taken as 30 lb/sq ft horizontally, or if very exposed, 40 lb/sq ft.

The stress in a headframe depends on the position of the hoisting engine, the height of the headframe, the depth of the shaft, the weight and capacity of the cage with full truck, or loaded skip. The average weight of a steel truck is one-third of the ore it contains; and that of a steel cage from 75 to 100 per cent of the total weight of its contents, depending on the class of ore. A steel skip weighs from one-half to two-thirds of the ore it holds. To guard against spillage, a skip should not be filled more than 80 per cent of its capacity. When hoisting in balance the strain in the two ropes is never the same, so considerable torsion is developed, first in one direction and then in the other. In countries where the weather is inclement, the headframe is housed in.

An A-frame (Fig. 35 (*d*)) is used for inclined shafts. It is simple, economical in material, and is easily braced to withstand vibrations. The sheaves being almost over the resultant of forces, they can rest on vertical posts.

The larger the sheave, the smaller the bending strain on the rope; but the inertia of a large heavy sheave with rapid hoisting may cause excessive wear on the rope towards the end of a hoist. With flat ropes wound on a reel, the rope is always in line with the headframe sheaves; but with a round rope, the horizontal or fleet angle between the sheave and the drum is constantly varying, so the distance of the engine from the headframe for a flat rope can be less than the distance for a round rope, unless a traversing drum is used. An ordinary hoisting engine should be so placed that the rope makes an angle of 40 to 45° from the horizontal so as to reduce the fleet angle.

Pyramidal Headframe. Each leg may be in one piece, or built up. When in one piece the sticks are round as they grew. If in sections the timber is squared. When selecting a tree of suitable height, note if there are any broken branches which may indicate that the tree is hollow, and if there is any doubt, drill a small hole to make sure. The diameter of the upper end should not be less than 9 in., and one must trust to Nature to make the butt of proportional strength. The height of the headframe may in some cases depend on the height of the timber available. The legs of such a headframe are connected in pairs before erection, the upper portion resting on a horse so as to give them a start when being raised into position. Each pair is placed at right angles to the hoisting

Gallows-frame, when of wood, is generally of squared timber. The position of the back-stays is determined by the parallelogram of forces. In Fig. 35 (*e*) let *a b* represent the ground line, *c d* the height of the headframe. As the force necessary to draw up *c d* must be at least equal to the winding force (say of *c e* equal to *c d*) draw *e f* equal and parallel to *c d*, the line *f c* is the resultant of forces. This resultant should always fall within the structure. If the back-stay is too near the shaft, not only is the bend of the rope too sharp over the sheave, but the structure is liable to be pulled over, especially in the case of a cage or skip sticking in the shaft. If the base of the back legs is within the resultant, the hoisting strain produces tension or uplift which is undesirable; if the base of the back legs is outside the resultant, they are under compression. If the back legs are too far away from the shaft they would not be strong enough to stand the weight of the load without extra support. So as to be on the safe side make *e′ c* equal to 2 *e c* and complete the parallelogram. Then *f′ c* is the line sought. The back legs or stays are braced and tied together. The distance the headframe is set back from the shaft depends on the diameter of the headframe sheave, the throat of which must come directly over the centre of its compartment.

Steel Headframes. The members may be tubular, box, or lattice, mostly the latter. The lattice type is built up of four angle irons joined together by short flat bars arranged diagonally, single or crossed with horizontal pieces between. When the angle bars are connected with plates they form the box type. The box type has more weight and offers more resistance to wind pressure than the lattice type, but is less costly to construct.

It is sometimes found necessary to make alterations to a headframe. If it is desired to have it a little higher, any addition has to be made at the bottom. This may be done by fastening strong beams of wood to the legs and using a series of jacks to press against them, gradually forcing the structure up. Other timbers are placed underneath to support the frame when the jacks have to be removed to prepare for another lift. To reduce the weight to be raised, all unnecessary parts are unbolted and removed.

A headframe may get out of plumb owing to ground movement. The method employed to rectify matters will depend on the nature of the ground and position of the solid rock. The headframe may be supported by reinforced concrete if solid rock is not too deep down, or on steel girders, one end of which is anchored to sound ground, while the other is supported by structural steel struts seated in solid rock.

If the ground will not allow the headframe to be erected directly over the shaft, it may be placed a little way back, and an incline made between the mouth of the shaft and the headframe.

Shaft Sinking

When sinking a new shaft, time and labour can be saved by sinking it in sections, if its site can be reached from underground workings. After careful and repeated surveys, a rise is put up from each level in the centre of the prospective shaft, and when the upper level is reached, the rise is widened out to the full size of the shaft by down holes, the broken material falling by gravity to the level below, from which it is trucked away. Instead of rising in the usual way, a bore hole may be drilled from the upper to the lower level for a rope to pass through. The rope is attached to a small skeleton cage from which a man can drill holes for excavating the rise.

When sinking a shaft, the men should be protected by a rock or timber pent-house, the winding being done through the area designed for a manway.

If while sinking the timbering has to be brought near the bottom, it must be protected during blasting by some form of shooting set or blasting shield. This may consist of a frame of *H*-beams to which 6 × 6 in. timbers are bolted, the whole being covered with $\frac{1}{2}$ in. perforated iron plates with the exception of a hole in the middle for the bucket and men to pass through. The perforations are for ventilation purposes. Chains at the corners of the shooting set are suspended from four strong chain blocks attached to the lowest set of shaft timbering.

When lowering the sinking bucket, it is prevented from swinging too far from side to side by a frame or jockey that runs on guides. When the jockey reaches the end of the guides, it rests on stops, and the bucket continues to the bottom slowly. When the bucket is hoisted again, a block attached to the rope about 6 ft above the bucket engages the jockey and both are hoisted up.

The lowest section of a shaft is provided with a chain or wire rope ladder which, being loose, is not hurt if hit by stones when blasting, and can be easily hauled up out of the way.

Supporting the Ground

Some ground stands well and requires no support, especially when the roof is arched. Drifts in a lode generally have full sets placed 5 ft apart. The caps have overhead laths to hold back mullock filling, while the sill helps when taking out the bridge later on. If a drift requires timbering, the timber should be examined periodically, and when a set shows signs of giving way, it should be replaced at once, otherwise the sets on either side, probably erected about tho same time with the same class of timber, will have to take more weight than they can stand, with the result that the whole drift may be lost.

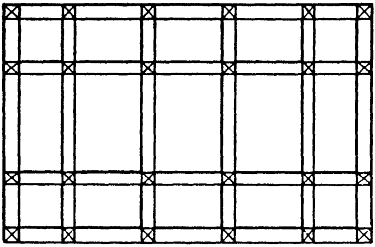


FIG. 36. PLAN OF JACKETED SHAFT

Certain ground which is firm when first excavated may become winded after exposure to the air and fret away. Such ground should be close timbered as soon as it shows its nature, before a large cavity is formed. The most difficult classes of ground to support are swelling and running ground.

Swelling ground is most likely to occur in serpentine rocks and certain shales and clay. Timber may have to be replaced several times before the ground becomes settled. When the squeeze has once been taken out of the ground, the timber may stand for years. The sets of timber in a drift break more frequently in swelling ground when the legs rest on sills. To avoid this, place pencil pointed legs direct on the ground, when the ground will swell round the timber instead of breaking it. Swelling ground should not be close timbered. Spaces between slabs should be left so that pressure can be relieved, either naturally by the ground oozing out, or it may be picked out before the timber breaks or becomes buried. In order to relieve side pressure on a rectangular shaft in heavy ground, the shaft may be jacketed, i.e. it is surrounded by a larger shaft, there being sufficient space between the two for men to work in when removing ground to ease the timber.

In each corner is a chute through which the excavated material is sent to the level below.

Running ground, as in some alluvial deposits, the roof, sides, and face, must be kept under control. Laths securely tailed, and supported on a false set a little higher than the main set, are driven ahead in the roof with bridging, but the face is kept from falling in by face-boards held in place by side laths. A face-board (Fig. 37) may reach right across the drift, or it may consist of two pieces which overlap in the middle where they are held by toms, so that only a small portion of the face

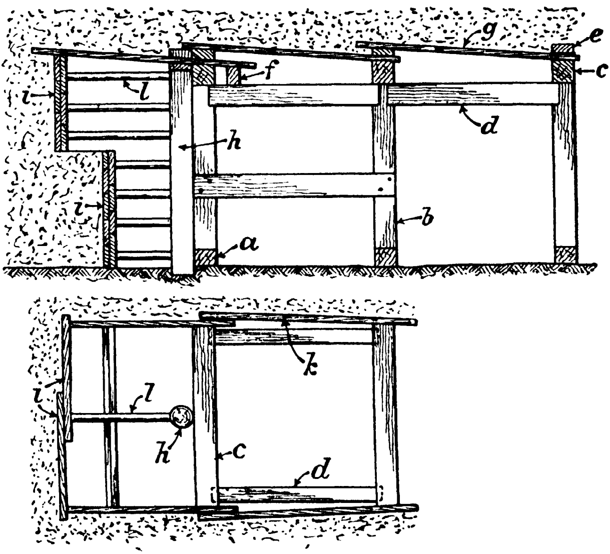


FIG. 37. WORKING RUNNING GROUND WITH FACE BOARDS

Top: Longitudinal section. *Bottom:* Plan

(*a*) Solepiece; (*b*) Leg; (*c*) Cap; (*d*) Studdle; (*e*) Bridge; (*f*) Tailboard; (*g*) Head lath; (*h*) Post; (*i*) Face-boards; (*k*) Side laths; (*l*) Toms

is exposed at a time. The toms are tightly driven between the face-boards and a post the top of which rests against the cap of the nearest set while the foot rests in a hitch in the floor. The top face-board, or half of it as the case may be, is removed, the required amount of material allowed to fall out, after which the face-board is replaced, the side laths driven up, and the face-board tommed with longer toms. The other face-boards are treated in a similar manner. When the advance is sufficiently far ahead, another main set is erected. If the post is inclined instead of being vertical, it will not be necessary to keep so many toms on hand, as those originally used higher up can be used lower down. The face- or breast-boards are planks 8 × 2 in.

In some cases shafts may be spilled or forepoled in a similar manner.

Shafts may be supported by timber, concrete, or iron, wholly or in parts. Vertical rectangular shafts have a plumb-bob hung in each corner. Circular shafts take their measurements from a central plumb-line.

If ground stands well, it may only be necessary to box it in to prevent any loose stones from falling. All timbering whether in shafts or drifts should be blocked and wedged firmly at all joints and the space between the timber and walls packed with waste. When a shaft is boxed with 2 in. planks, a space of 2½ in. should be left between every five sets so that boards can be inserted to form a platform when necessary to make repairs. Box timbering is built from below upward from bearers in 30 ft sections. Frame sets are used for large shafts and where box timbering would not be strong enough. Vertical slabs are used at the back between the sets. If the ground is very heavy, it may be necessary to use solid cribbing. Frame sets are generally placed from above downward, each set being suspended from the one above by 1 in. hanging bolts with large washers. Each hanging bolt is made in two pieces which are hooked together. They are made 4 in. longer than required to allow for adjustment. Being in two pieces, these bolts can be readily removed and used elsewhere when the timbers are once firmly fixed in position, which would not be the case if the bolt was in one piece.

When required to replace a frame set, deck over a set above the one to be replaced so as to protect the men at work, also deck the set below for the men to work on and also to prevent material from falling down the shaft. If the frame set to be removed is suspended by rods, remove them. Cut out a slab,

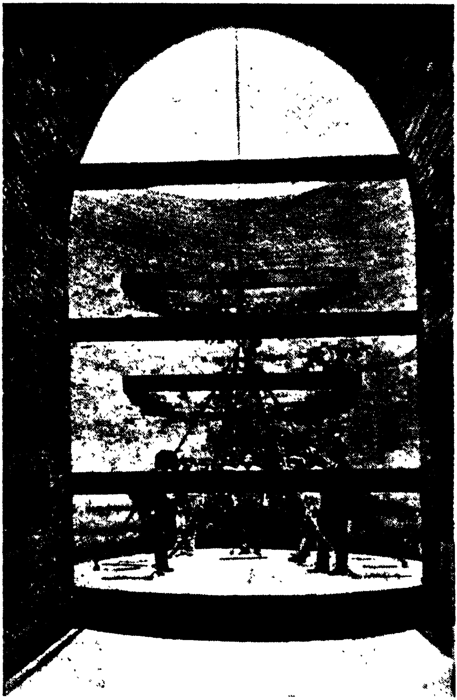


Fig. 38. Walling Stage

remove the packing at the back, take out the rest of the slabs, packing and blocking. Knock back the studdles. The dividers are next removed. A wall-plate from one side is taken out and sent to the surface, then the other, and these are followed by the end pieces. The new sticks are now lowered and placed in position in reverse order. If there is not room to swing a wall-plate on account of the dividers of upper sets, it may have to be made in two pieces, the joint being under a studdle.

Concrete and brickwork lining is generally 18 in. thick. It offers less resistance to ventilation than timber. It costs more, but the fire risk is eliminated, and there is saving in maintenance and inspection. If the ground through which a shaft is sunk is watery, as in some sandstones, holes are drilled in a short distance and cement pumped into them. If water comes through cracks, plug the cracks except here and there at places where cement is pumped in.

For lining circular shafts with bricks, a walling stage (Fig. 38) is used for the men to stand on. It also serves to protect any men working below from material falling on them, and the lining from damage by blasting. It consists of a frame of angle iron decked with planking and protected underneath with plate iron. The stage is partly fixed and partly movable so as to allow it to pass between buntons. The stage is suspended by ropes with bridle chains and is lowered or raised by powerful engines geared for slow speed from 16 : 1 to 28 : 1. The staging is generally 6 in. smaller in all directions than the inside of the shaft, and when in use, it is steadied by small sprags between it and the side of the shaft. An opening is left in the middle of the stage provided with a trap door to enable miners to reach their work and send the excavated material to the surface in a bucket; there are also holes for air and water pipes to pass through.

Iron tubbing is resorted to in circular shafts when the pressure of water is too great for bricks. The cast-iron tubbing is made in sections convenient for handling and of such sizes as to fit the shaft when put together. The vertical joints break joint in each course. Soft wood sheeting is placed in both the horizontal and vertical joints. In the English method of tubbing the flanges, cross-ribs, and brackets are all placed at the back of the tubbing, it being found that the pressure on the outside together with the wedging is sufficient to retain the segments in position. The segments are from 18 in. to 3 ft high. A hole in the middle of each segment allows water to pass through while placing the tubbing. If the watery stratum is not too thick, a bed is levelled when solid rock is reached, and a cast-iron wedging curb laid down, on which the tubbing is built; but if the stratum is thick, intermediate curbs may be put in at different levels and the tubbing completed in stages. If water rises above the top course of tubbing, it is sealed by a wedging curb. The horizontal and vertical joints are then carefully wedged, and as each section is finished the holes are plugged.

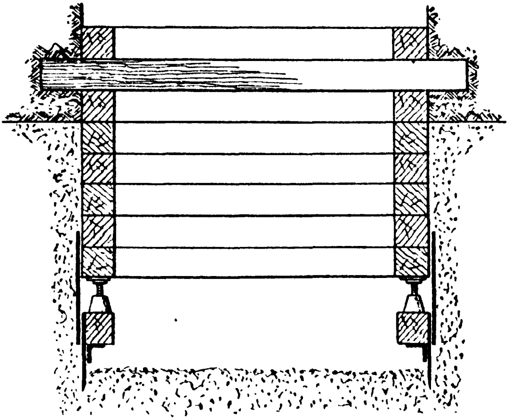


FIG. 39. SINKING BOX

In rectangular shafts of moderate size if loose material is encountered, a sinking box is used. This consists of steel plate ¾ to 1 in. in thickness according to the size of the shaft, and 3 to 6 ft deep. The bottom has a cutting edge. The several plates are butted together and connected with vertical covering plates, 12 in. wide by 1 in. thick, which are riveted on to one plate, and bolted to the other when put together. The bolts and rivets are ⅞ in. diameter. At the corners, the side plates are connected with narrow end-plates, about 2 ft wide, with 4 × 4 in. angle irons. The centre piece of the end is bolted to the two corner pieces by means of 4 in. angle irons which are riveted to the side of each plate. A 4 in. angle iron is also riveted all round the inside of the box which strengthens

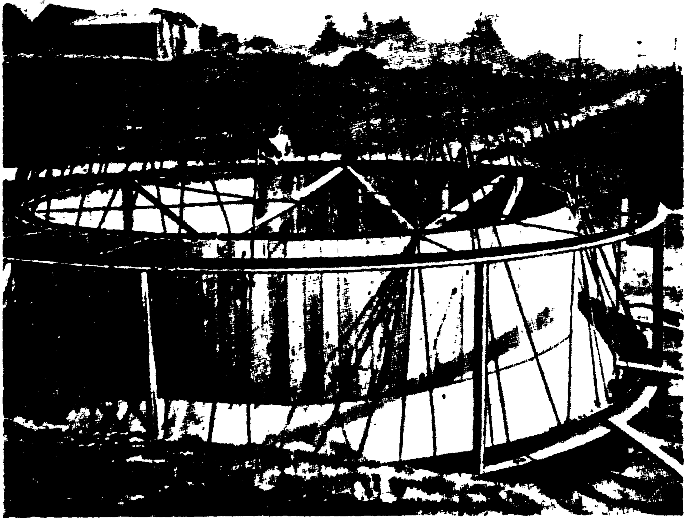


FIG. 40. BOTTOM SECTION OF A CONCRETE DROP SHAFT

it and serves as a support for timbers 16 × 6 in. to which it is bolted. On these timbers are placed screw or high pressure hydraulic jacks that force the sinking-box down. To the top and back of this box are bolted ½ in. plates 3½ ft deep. These upper plates are butted close together and stand up behind the lining of the shaft to prevent the drift from finding its way between the sinking-box and the timber of the shaft.

The sides all complete, with corner plates forming the ends, are put together at the surface and sent below, where they are bolted to the centre end-piece. The box is driven down for a depth of not more than that of a set of timber at a time. The jacks are run back from one side of the shaft; a new wall-plate placed in position and held there by jacks between it and the top of the drift-box. The other wall-plate is dealt with in a similar manner, and finally the end-pieces are put in place. Running ground may be retarded to a certain extent by partly draining the ground ahead through a bore hole.

Drop shafts (Fig. 40) are put together on the surface and lowered as sinking proceeds. They are made of wood, concrete or iron. When the loose stratum is passed through, a joint must be carefully made between the drop shaft and the regular lining. When concrete is used, the shoe consists of two steel plates ½ in. by 20 in. which are riveted the at bottom and flaired at the top to take in the lower part of the concrete wall. The bevel has an angle of 55°. The concrete is connected with the shoe by ¾ in. 8 ft bolts, the heads of which are countersunk. The forms may be of wood or iron-plate. The thickness of the concrete does not depend entirely on the pressure it has to withstand, but also on the weight which is depended on to enable it to sink. It may be as much as 30 in. thick. The concrete is reinforced with 1 in. iron rods arranged vertically. Where the rods end they overlap slightly. This reinforcement is necessary in case the upper portion of the lining should stick, when the lower portion might break away. No horizontal reinforcement is required. At first 5 ft of 1, 2, 5 concrete is laid and allowed to set for a week, after which the lining is built up in 10 ft sections when the lower one is sufficiently hard. Circular drop shafts are easier to sink than rectangular, and owing to arching action, their walls can be thinner.

Rectangular shafts require less excavation for the same useful area, but the walls must be thicker for the sake of strength. Great care is necessary when sinking in quicksand to maintain the bottom edges exactly level and so prevent the shaft from getting out of plumb. It is also necessary not to hoist more sand at a time than actually required, as fresh sand would run into the cavities formed at the back, and falling suddenly might cause the lining to break. If unusual bubbling is noticed, a plank is pushed down and water may be allowed to rise, say 15 ft, and sand is lowered from the surface to steady the shaft. After a few days, when everything has had time to settle, the operation of sinking may be continued. If ground caves, the caisson tends to lean towards it. To remedy this, cut a trench on the upper side of the caisson and pile up dirt against the other side. If a boulder obstructs the descent of the shoe, it must be blasted out. Should the caisson stick in mud, agitate the mud by blowing compressed air into it through 1½ in. pipes.

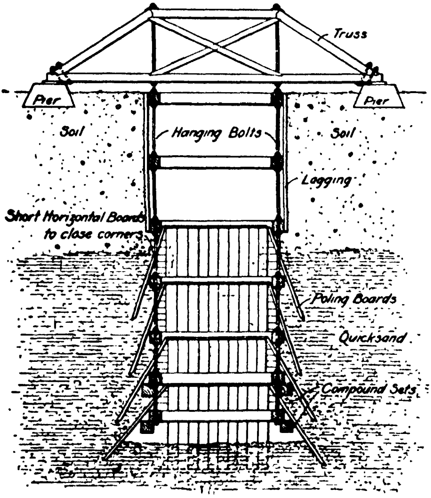


FIG. 41. RECLAIMING A FALLEN-IN SHAFT

Concrete lining for shafts may be either in the form of precast bricks, or better still, as a uniform monolith, being stronger and making a better seal should water be encountered. If there is water, short pieces of piping must be left in the concrete to drain it till the concreting is finished, when they are closed with gate valves.

If a monolith lining is required for a section of a shaft, a ring of horizontal holes is drilled into which old drills are wedged, projecting into the shaft. On these is laid a timber curb on which the first ring of iron-plate forms rest. The concrete is prepared on the surface and lowered in bottom hinged buckets. If the shaft is large, the concrete may be lowered to a distributing box from which it is conveyed by four large rubber tubes to the back of the forms.

Repairing a Fallen-in Shaft. It is sometimes necessary to repair a shaft, the mouth of which has fallen in, owing to the timber rotting, or having been burnt out by a bush fire (Fig. 41). If the funnel-shaped hole is small, it is bridged over by two beams, or if large by built-up trusses. A platform is placed on these for men to work on. Hanging sets are suspended by iron rods over the shaft, the dirt that has fallen in is raised in buckets and thrown behind the lagging to fill the cavity from which it came. When solid rock is reached, bearers are put in. As it is not known when the debris in the shaft may give way, the man cleaning it out should have a rope round his waist, while the other end is fastened at the surface (Fig. 40).

Safety Precautions

Where natural strata are not safe, the shaft should be securely cased.

When sinking another section below the lowest level of a mine where work is in progress, a suitable penthouse should be provided to protect the men who are sinking.

Guides should be used in shafts where men or mineral are hoisted. If the hoisting is done in buckets, a jockey or crosshead should be employed to prevent the bucket having too great a swing.

When men are raised or lowered in a bucket, they should wear safety belts attached to the ring of the bucket by a chain or rope.

A competent person should at least once in twenty-four hours examine the state of all safety appliances or gear connected with the shaft and cages, and any necessary alterations should be immediately attended to.

If work is carried on at night, the surface at the top of the shaft should be sufficiently lighted.

Ladders. Every shaft should have a ladder compartment securely separated from the adjoining hoisting compartment. There should be sollars not more than 30 ft apart. The ladders should be arranged, one below the other, like the slates on a roof. They should be at a suitable angle so that a man climbing them would be approximately in a vertical position. The rungs of the ladders should be at regular distances apart, about 10 to 11 in., depending on the angle of the ladder. The inside space between the sides of the ladder should be 12 to 15 in. Ladders must be firmly fixed top and bottom to prevent them slipping or swaying, and where two ladders are joined together, they should be stayed. If a ladder does not project above the upper man-hole, hand grips should be provided. In cases where a ladder has to be fixed vertically, it should have two points of suspension, and shall either be fixed in a corner or have 4 in. blocks attached so as to keep it off the wall and give sufficient toe room.

Cages. Every completed shaft exceeding 60 ft, where engine power is used for winding, shall be provided with a cage.

No safety cage should be used till it has been tested and certified to be in good condition.

Cages and skips should be provided with suitable safety catches where winding is not so fast as to cause the catches to act under ordinary conditions. Where winding is too fast for the use of safety catches, a greater factor of safety must be employed. All safety catches should be tested from the drum at least once in every month, and should be set to act within 2 or 2½ in. drop.

Every cage should be provided with some means to prevent skips from projecting out of the cage. If men travel in a cage, the two sides must be closed to prevent any part of the body projecting out of the cage. An inclined roof of steel not less than ¼in. thick should be hung on hinges capable of being opened from inside. The roof should be at least 6 to 6½ ft above the floor of the cage.

Tools, timber, pipes, rails, explosives, and other material should not be carried in the same cage as any person. This does not apply to timbermen, pitmen, surveyors, and their several attendants.

Keeps or chairs for supporting a cage should be provided at all places where a cage may be required to rest.

The banksman or braceman should have command of shaft signals. He should see that not more than the specified number of persons is allowed to ride in the cage.

The onsetter underground should have full control of signals when communicating with the surface. He should see that the skips are properly placed and secured on the cage before signalling. The onsetter should not allow men to get on or off a cage while it is in motion.

A notice should be kept in a prominent place on the surface and on every plat underground stating the number of persons who may ride in a cage at the same time, and the man in charge of the cage should not allow any excess of that number to enter the cage.

When buckets are used for lowering material, the banksman should see that the bucket is properly loaded; that no loose material is packed above the level of the bucket; that gear and tools are placed in an empty bucket, and if they project above the top are securely fastened to the bow or chains of the bucket; also that timber or other bulky articles are safely slung.

Deciding the Most Suitable Method of Extraction

There are several points to be carefully considered before deciding on the most suitable method of extracting ore under existing conditions. Later on, with fuller information and with changed circumstances, it may be advisable to alter the method originally adopted. A method of extraction suitable for the upper levels of a mine may be unsuitable in the lower levels. Most of these points require local knowledge.

1. The Deposit.

(*a*) The physical character of the ore; whether it will stand without support. Occasional slippery heads or loose blocks may require temporary support.

(*b*) The shape and dimensions of the deposit. It may be necessary for a rich narrow vein to have some of the wall excavated so as to make room to work in. This rock may be excavated separately to the ore by resuing, which not only keeps the ore cleaner, but provides filling on the spot, thus saving much handling. In narrow veins where the walls stand well and no waste is made in the mining, stull-pieces are placed between the walls at regular distances to support staging for miners to work from.

(*c*) Whether the values are regularly or irregularly distributed; the size and frequency of ore shoots.

(*d*) The angle of the deposit: whether more nearly approaching the vertical or horizontal.

(*e*) Whether the walls are regular or irregular.

(*f*) The position of the deposit, whether above or below the surface water level; also the depth of the overburden, if any. Whether the ore can be won by open cuts, or must be worked from underground passages. If near deposits, which bound a large block, are worked simultaneously, the block may collapse and do much damage.

(*g*) The immediate and probable future demand for the ore, and whether the demand is likely to be continuous or intermittent.

(*h*) The amount of development work likely to be required.

2. The physical character of the wall-rock, especially that of the hanging wall. Whether it is strong enough to stand without artificial support, or whether it is soft and liable to fret away when exposed to the air. Whether systematic timbering or filling must be adopted.

3. The Means of Support.

(*a*) Whether the deposit is of such value that one can go to a fair amount of expense in obtaining outside material to secure the workings, or whether the value is so small that it is cheaper to leave ore behind as pillars.

(*b*) Whether there is sufficient waste mined or available at a reasonable cost to fill the excavation, or whether there are barren portions that may be left for support. Filling may be obtained from development work, old waste heaps or tailings, from the excavation of surface water tanks, or may be specially quarried.

(*c*) The nature and cost of timber available for mining purposes. Timber can only sustain a certain amount of weight, and is liable to be destroyed by fire, or attacked by certain insects, and fungi, so should be looked on as a temporary expedient.

4. If the surface is of such value or of such a nature that it must not be allowed to subside after the mineral has been extracted.

5. The water available and its pressure, either for power, hydraulicing, or dredging. The amount of water to be handled in underground workings.

6. The cost of supplies and power.

7. The type of labour available.

8. Whether drilling is to be done by hand or machine.

9. Ventilation problems.

10. Financial matters have to be taken into account. A mine that has to pay its way from the first with the assistance of little or no capital, may be forced to use certain methods, such as picking out the eyes of the mine, although the manager may know it is a bad thing to do and uneconomical in the long run.

In some secondary enrichments, bunches of ore are found having little or no connection with one another. Also in deep alluvial deposits, some parts are richer than others. It has been proved that it is better to work deposits systematically instead of in an irregular manner, as it is easier, and there is no fear of missing rich portions which might otherwise be lost.

Deposits are generally worked in benches, steps or stopes. These not only supply several faces from which ores can be obtained simultaneously, but by breaking up a continuous line, which would be a line of weakness, there is less chance of the ground collapsing.

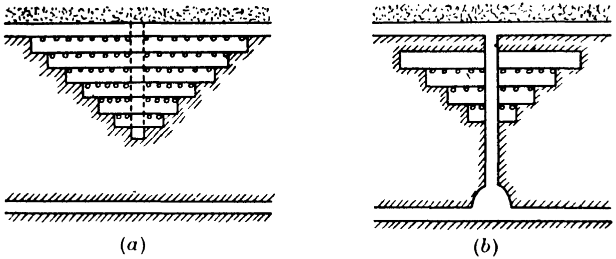


FIG. 42. UNDERHAND STOPING
(*a*) Without a winze; (*b*) Winze and pillar below upper level

The two main forms of stoping are underhand and overhand, both of which have various modifications. It is cheaper to mine ore by stoping than it is by driving or sinking. When the ore is worked out of a stope, the empty place is called a "stull," and any timbers placed from wall to wall are called "stull-pieces."

If two small veins or beds occur close together, they may be worked from one stope, but should a band of rock between be too thick, it may be necessary to work each separately. If two veins with a flat underlie are worked together it is usual to commence work on the upper vein, unless the roof is bad, otherwise it might be difficult to reach the upper one.

Underhand Stoping (Fig. 42 (*a*) (*b*). This method is useful when an ore body does not extend to the level below, but if the body is long, it is better to put in an intermediate level and overhand stope so as to save handling. With irregular deposits underhand stoping may save much unnecessary development work for deeper levels.

The advantages of underhand stoping are—

(*a*) The ore can be extracted at once without waiting for a shaft to be sunk deeper and a lower level driven underneath the point selected for the stope.

(*b*) The work of drilling and picking is easier than with overhand stoping, as they are all down strokes.

(*c*) When the roof and loose material are secured, there is less danger of ore or rock falling on the men than with overhand stoping.

(*d*) There is less loss of rich particles of ore than when overhand stoping.

(*e*) If the mine should have to be abandoned, the cost of driving a lower level would be saved.

(*f*) The men standing on solid ground are not liable to staging accidents.

If two levels are connected by a winze, there are further advantages—

1. The ventilation is improved.
2. There is an extra passage through which men can escape in case of an accident.
3. Any water is got rid of without pumping or bailing.
4. The hoisting of the material to the upper level is avoided.

The disadvantages are: Much timber for stull-pieces and planks is required, and has to be left. If mining has to be suspended for a length of time, due to war, strikes, or want of a market, and the timber gives way, the debris falls on the surface of the ore. If levels are not connected by a winze, the labour and expense of raising the material by hand is a great drawback. Water may accumulate on the floor and make it unhealthy for the miners. If two levels are connected by a winze, as the floors of the stopes are given a slope towards the winze, loose ore may fall on the men working a lower slope, or the men themselves may slip into the winze.

Overhand Stoping (Fig. 43). Though the ore is broken down from overhead, it does not follow that all the holes are uppers, for some are breast and some down holes. Miners have a tendency to drive stopes longer horizontally than to rise vertically. Stoping is generally started from the bottom of a winze, but this is not always the case (*a*).

Some of the advantages of overhand stoping are: There is no hoisting of ore or water to the level above, as they pass to the lower level by gravity. When timber is required, much of it may be recovered. If the workings are temporarily abandoned, they will later on be found in better condition for future work than underhand stopes. Gravity assists in breaking up the ore as it falls. Ventilation is easier. It is generally found more economical than underhand stoping.

Disadvantages are: There is greater danger of falls of ore from the roof than in underhand stoping; but the miner being nearer the roof than when underhand stoping can sound baulky ground, and either break it down or support it. When upper holes are required, they are more inconvenient than breast or down holes. With overhand stoping there is more loss of fine material by becoming mixed with filling, but this may be obviated by laying down old canvas or bagging under a plank flooring on to which the ore is broken.

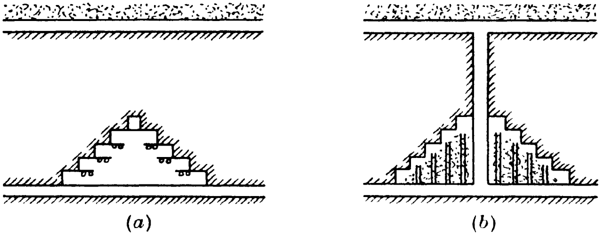

(a) (b)

FIG. 43. OVERHAND STOPING
(a) Without a winze; (b) Flatback with winze

For flat lodes, the ore may be mined by underhand stoping from above and overhand stopes from below at the same time, which eventually meet.

There are many varieties of overhand stoping, such as Flatback, Rill, Sub-level, and Pillar and Room stoping.

Flatback Method (Fig. 42 (*b*)). This is a cut and fill method. The stopes, which are generally started from the bottom of a winze are carried horizontally. The first stope is called the leading stope; subsequent stopes are known as swinging stopes. If a horizontal pillar is not left above the drift, the leading stope is started from the drift. Though the bridge or horizontal pillar below the upper level may be strong enough for the length of one stope, it may not be strong enough when the stopes from both sides meet, so when approaching the bridge, stoping on one side should be completed before the stope from the other side reaches the bridge. In cases where the ore is friable, a modified square set method can be combined with the flatback. Sills are laid on the filling and round tapered legs are placed on these, caps rest on the legs, and laths placed over the caps. When the next slice is to be mined, the laths and caps are removed and the legs pulled out for re-use, only the sills being lost. The new sets are placed between the sites of the lower sets, so as to permit of the old legs being withdrawn.

Additional advantages of overhand stoping, besides those mentioned above, are—

(*a*) Machines are readily rigged up.

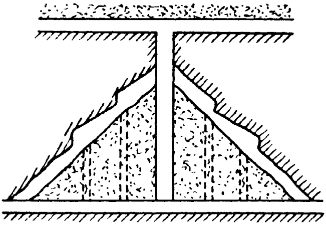


FIG. 44. RILL METHOD

(*b*) Doubtful pieces of roof are easily supported.

(*c*) Irregularities in the walls do not affect the method, and the stopes can be contracted or expanded to suit the deposit.

(*d*) Poor ore, or horses of mullock if small, can be shot down and left for filling, or if large can be left in situ.

(*e*) Ore can be picked in the stopes and waste left for filling. Ore can be drawn as mined, so no capital is locked up as in the shrinkage method.

(*f*) Supplies are readily carried through the stopes.

(*g*) Filling in the flatback method being laid down in comparatively thin layers, it packs more tightly than when laid down on the rill as there is more general traffic over it, and the weight of the ore broken down helps to make it more compact.

(*h*) The absence of timber among the filling, as when square sets are used, is an advantage as it is impossible to pack tightly under each cap and strut.

Disadvantages are: All the ore has to be handled, and the filling has to be spread which makes it more expensive than the rill method; also frequent ore passes and man-ways have to be built.

The Rill Method (Fig. 44). Suitable for narrow or wide lodes with a steep underlie: if less than 40° much shovelling is necessary to assist the rilling. The method is commenced as a flatback, with temporary passes, till of sufficient height to start rilling. Rilling commences from the top downward. A leading stope is carried out on both sides of a winze at an angle slightly flatter than the angle of repose of the filling—say 33°—so that the latter will not start running easily when men climb the stope, or a machine is rigged up on it. The ore of the stope on one side is broken down while the other side is being filled. Ore passes are built about 20 ft apart, and are raised about 3 ft above the filling, to prevent it diluting the ore. The face of the stopes is generally made saw-tooth shape so that all holes can be down or wet holes. When stopes from adjoining winzes meet, all the filling can be rilled into the pass at their junction. Subsidiary winzes are sunk in the wedge-shaped block of ore that is left through which filling can be dropped. The stoping ends about 25 ft below the upper level, the filling squared off, and the bridge extracted by any suitable method.

The advantages are—

(*a*) All holes are down holes.

(*b*) Most of the filling is done by gravity, thus saving trucking or scraping in the stope.

(*c*) The shape of the roof prevents horizontal slabs from breaking off as they are cut at an angle.

(*d*) The ore is filled out quickly so there is no fear of sulphides heating, and capital is not locked up as in the shrinkage system.

The disadvantages are—

(*a*) It is not adapted for very irregular deposits.

(*b*) Ore cannot be sorted from the waste.

(*c*) The attention of the miner is partly focused on the men working below him.

(*d*) It is more difficult to support the roof from a slope of loose material than in a flat back stope.

(*e*) A place has to be prepared for a drilling machine; also when a crib is required, and the top of a crib does not come square against a sloping roof.

Shrinkage Stoping (Fig. 45). This can be used for deposits the walls of which stand well, are fairly steep and regular. The actual breaking of the ore may be by flat back or rill. Miners stand on the broken ore to reach the face. Some 60 per cent of the broken ore is left in the stope, the balance is drawn so as to leave room (4 to 8 ft) for the men to work in. It is specially useful for isolated deposits where there is no level above along which filling can be brought, and it does not matter if the walls collapse after the ore has been extracted. The broken ore is drawn from one or both ends of the stope, but chutes can be inserted anywhere along the drift if required. There are passes at both ends to act as travelling ways and for ventilation.

Its advantages are—

(*a*) The method is safe.

(*b*) Only a little of the hanging wall is exposed while ore is being stoped.

(*c*) The roof overhead can be easily sounded.

(*d*) Mining is continuous as there is no space to fill.

(*e*) The ore in the stope is a reserve in case of trouble with workmen.

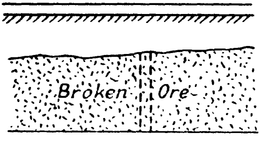


FIG. 45. SHRINKAGE METHOD

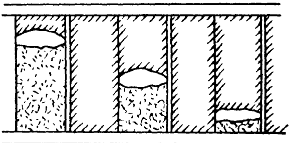


FIG. 46. ROOM AND PILLAR

(*f*) The ventilation is good as the air is obliged to travel past the face, so clears the fumes from blasting quickly.

(*g*) Little development is required, and no passes have to be built.

The disadvantages are—

(*a*) All material broken must be taken, as sorting cannot be done in the stope.

(*b*) It is not suitable for ore that is likely to heat when broken, or to oxidize—if harmful for subsequent treatment.

(*c*) A large amount of capital is locked up till the ore is drawn.

(*d*) If the deposit has irregular walls on which ore can rest, it may be shaken down, by blasting in a neighbouring stope, on to men who are cleaning up the floor.

(*e*) Branch veins cannot be worked by this method.

(*f*) It is not suitable for narrow lodes where moisture may convert the ore into a compact mass.

As big pieces of ore are bound to be mixed with those of average size unless the miners are carefully supervised, it is usual to have a bulldozing drift over the trucking drift, where a grizzly of iron rails is placed, and pieces of 10 in. or over are broken up.

Room and Pillar (Fig. 46). Used for wide low-grade lodes where the wall pressure is too great for the whole width to be mined at once, so it is divided into rooms and pillars, either in the direction of the strike, if not too wide, or transverse. The rooms may be worked out on the flat-back, rill or shrinkage system. The rooms and pillars may be of the same size, say 50 ft, or if the ore stands well, the pillars may be narrower than the rooms. The width depends on the arching ability of the ore. The boundary between the room and pillars on either side is timbered to prevent any filling from falling into the workings of the pillars later on. At the junction between a room and a pillar a winze is sunk which will serve both. If the ore is very low grade, it may be considered cheaper not to mine the pillars. As to how the pillars are mined depends on the physical nature of the ore. The core of a pillar may be stoped out by shrinkage stoping, and then by undercutting the base of the remaining pillar to cause a cave over a large area; or by making a series of drifts and cross-cuts under a pillar and blasting down what is left, causing the pillar to cave.

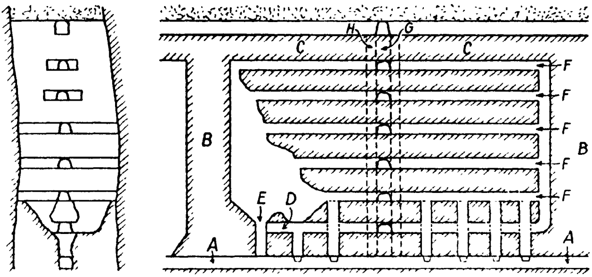


FIG. 47. SUB-LEVEL STOPING

A, Hauling level; *B*, Pillar; *C*, Floor pillar; *D*, Grizzly level; *E*, Mill hole; *F*, Sub-levels; *G*, Man-way pillar; *H*, Man-way rise

The advantages and disadvantages are those of the method adopted for extracting the ore. The method is flexible as the width of rooms and pillars can be varied according to how the ore will stand. The loss of ore in the pillars is generally great.

Sub-level stoping (Fig. 47). This is an open stope method, the primary requirements of which are that the ore and walls stand well and that the underlie exceeds 45° so that the ore will run freely along the footwall. Sub-level mining is not suitable for places where buildings on the surface must not be disturbed. Where conditions are suitable, this method being cheap, is good for low grade ore. The ore body is divided into rooms and pillars. If under 75 ft in width, the stope is taken out longitudinally, if over 75 ft it is taken out transversely from wall to wall. The distance between haulage levels is longer than usual, say 250 ft, which will leave room for five sub-drifts 35 ft apart and a level pillar of 75 ft in which conical chutes are excavated, and the ore has to pass through grizzlies with bars 20 in. apart, the oversize being broken up. The different sub-drifts are reached from a rise put up in the footwall, or from a shaft if conveniently situated. The sub-level drifts are 7 ft by 5 ft. Whether they are driven in the middle of a horizontal pillar, or at the sides, depends on the width of the block to be mined. Mining commences on the first sub-drift above an undercut, so each successive sub-drift is protected by the one above it. Ring drilling is used, about 24 holes radiating from the centre of the drift. The holes vary in depth, depending on their position and whether they are drilled by percussion or diamond drills. Diamond drills can bore up to 66 ft in depth if required. The cost when diamond drilling is used is less than with percussive drilling as, the holes being longer, the distance between the sub-drifts can be greater. Long sectional steel rods are used in 24 in. runs, the longest piece being 6 ft. They are made of 1⅛ in. round nickel-chrome steel. The couplings are heat-treated vanadium-steel, 2 in. long and 1½ in. outside diameter, a shallow thread being used.

When diamond drills are used in hard ground, they make less dust than steel drills, and if they have solid crowns, there is no time wasted in extracting the core. The rings of drill holes are 4 to 5 ft apart according to the nature of the ground. Blasting is done at the end of the week so as to allow plenty of time for the fumes to clear before Monday. Delay fuses and electricity are used for firing. Open stope, filling and pillar mining follow each other in regular order.

The advantages are—

(*a*) Drilling and blasting are done in drifts where the men stand on solid ground and are protected overhead.

(*b*) High blasting efficiency is obtained as the holes drilled are laid out to obtain the lowest combined primary and secondary breaking costs, which results in better and cheaper work than when the location of each hole is left to the judgment of the drill runner.

(*c*) There is no need to handle explosives daily.

(*d*) Drilling delays due to blasting, barring down, and mucking are eliminated, so drilling can be continuous.

(*e*) The drilled-off ore is practically a reserve of broken ore.

(*f*) There is no fear of fire.

Breast Stoping (Fig. 48). This is used in narrow veins. When the underlie is flat—less than the angle of repose—the deposit is mined with a long face, somewhat like the longwall system of coal mining. This is known as the longwall stope method. In such a case, trucks, or shaking troughs, can be brought close to the face for removing the ore. If the underlie is steep, the face is worked on an incline from one level to the next above and is known as a side stope. In this case the filling is tipped into the empty stope as close as possible to the breast. The disadvantage of side stoping is that it can only be employed where the walls will stand sufficiently long before filling takes place. It is more dangerous than other methods, as stone may fall on men working lower down. The levels must be closer together, which means more development work for a narrow vein.

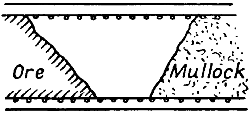


FIG. 48. BREAST STOPING, SIDE-STOP METHOD

Excavating the Bridge

A bridge or arch is that strip of ore between the top of a stope and the bottom of the level above; generally 25 to 30 ft thick depending on the strength of the ore and width of the lode. In wide lodes it is more expensive and dangerous to extract a bridge than when narrow, as the bridge has not the same advantage of arching action. When mining a bridge, it is advisable to drill shallow holes by hand as machines might cause too much vibration; also to use light charges. The cost of mining a bridge may be greater than the value of the ore, in which case it is left in situ. In a wide stope it is usual to prepare a sill floor before introducing the filling so as to make it easier when coming up from below to hold back the filling. This is done by placing sills parallel to or at right angles to the strike. Across these sawn timber or poles are placed close together, breaking joints. If the timber is likely to rot before the bridge is mined, there is no object in going to the expense

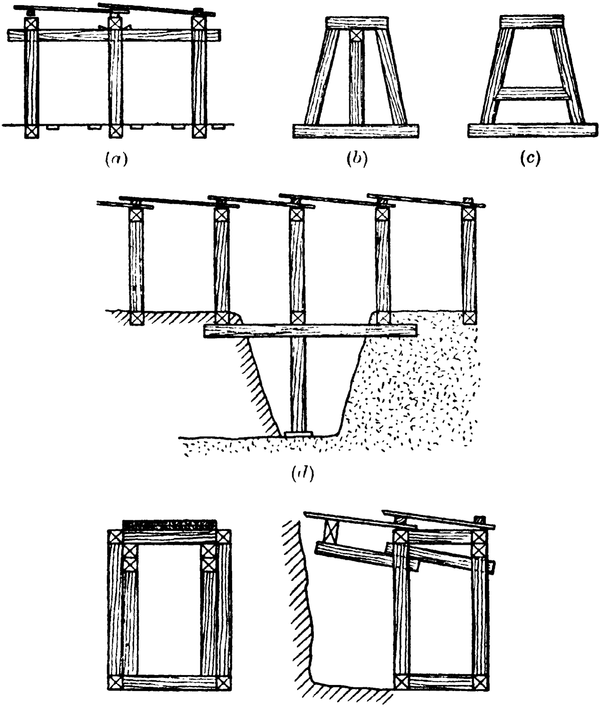


FIG. 49. EXTRACTING BRIDGE UNDER DRIFT

a) Longitudinal section showing boom supported by post which passes under three caps. The ground under two has been extracted. The far end of the boom supports the cap of the set to be undermined. (b) Cross-section of full set showing post supporting the boom. (c) Cross-section showing sprag to press legs of set to be undermined against the walls. (d) Another method of supporting a set while the ground is being extracted. (e) Cross-section. (f) Longitudinal section.

of laying down the timber. The bridge may be mined in horizontal or vertical slices, leaving the smallest possible space unsupported at a time.

When dealing with narrow veins, the bridge is taken out in vertical strips. If the upper drift has been timbered with full sets, these must be secured to avoid collapse when the floor is removed. This is done by placing a sprag between the legs and wedging them tightly against the walls. A boom, long enough to pass under the caps of three sets, is lifted into position and held there by a post under the middle cap. The boom is then wedged tightly against the three caps. When the ground is removed from under the sill of the set suspended by the boom, the sill naturally falls away, but can be replaced after the ore has been extracted and the ground filled in. Sometimes a post is placed under the sill as an extra precaution. As filling replaces the ore that has been extracted, the boom is pushed forward to another set, and the process is repeated.

When extracting the bridge of a wide lode, the sets in the drift are secured in a similar manner to the above. The top of the stope is levelled to within 5 ft of the back which is supported by bulks or pigsty timbering, properly arranged and wedged, leaving a gangway connecting ore passes. The bridge is taken out from wall to wall.

If the ore is friable or broken, it is better to work it out in vertical strips by hanging square sets (Fig. 50). This is started from a winze or short square-set rise put through the bridge. This method has the advantage of underhand stoping; the men are protected overhead when the backs are bad, only the top sets require to be lagged over, whereas if overhand stoped, back laths would have to be driven for every floor. The first strip is taken out, next the winze or rise, and when completely timbered, it is used as a pass, and the winze is filled in. Proceeding in this way, only two strips are open at a time; one used as a pass, and the other being worked. It is advisable to have many faces going at once, as the removal of the bridge is often a race against time, so as soon as the first strip is sufficiently far ahead, strips on either side can be started which are worked in a similar manner.

The leading stope is taken out under the sill floor of the level above, and when sufficient ground has been excavated, a full set is erected, and on the sills of this close to the legs are two posts on the top of which is supported a boom, from 6 by 6 in. to 10 by 10 in. If there is not room for a boom 10 ft long, a composite boom made of two pieces 8 ft long is used. The tail of the boom is blocked firmly against the back cap. The front end of the boom supports a head block over which 8 ft spilling laths, 10 by 2 in., are driven in an upward

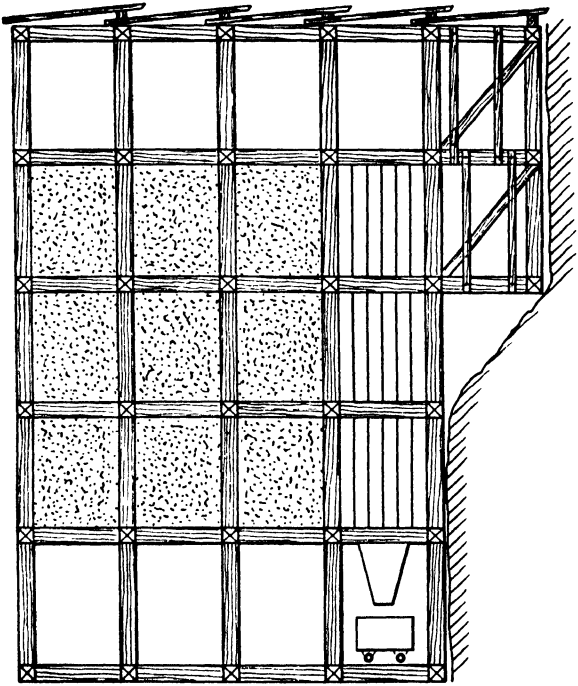


FIG. 50. HANGING SQUARE SETS

direction which serves as a protection to the miners while they work out sufficient ground to enable them to place the next set of timber. The sills are firmly blocked and wedged against the face, and 10 by 4 in. diagonal struts are placed between the legs of adjoining sets to keep them against the face. Eight ft hanging laths 5 by $2\frac{1}{2}$ in. are nailed to the sill and cap to suspend the sill while the ground is being worked under it: two hanging laths are used on each side. These laths really have very little weight to support as the wedging and blocking are sufficient to keep the sill in position until it becomes the cap of the set below. The booms are kept in place till the run of ground below is worked out, when the timbers rest on the filling below and the booms are moved forward to the next set. When the first row of stopes have reached their limit, wing stopes are started alongside the first row. If the ore is very loose, it may be necessary to use side laths and breast boards. The strip used as a temporary pass is lined with slabs 5 ft 9 in. by 10 in. by 1 in.

If the upper stope to be picked up has been timbered with square sets, and a rise has been put up in advance, the work is really an amplification of such rise. One set in the upper stope is temporarily supported by long booms reaching to adjoining sets, after which the ore below is removed and the timber itself supported from below. When a rise is not carried through, one cannot guide the posts of the lower stope to come directly under those of the upper stope: it is for this reason that long sills are used under the timber to be picked up. The set it is decided to undermine first is supported by long timbers resting on posts which stand on sills. Wedges are driven tightly between the long timbers and the cap. The ore is then carefully removed. When the posts of the upper set do not come over the posts of the lower set, the sill immediately beneath each post of the upper set may be supported by four braces, the bottoms of which are placed at the lower four corners of the bottom set in such a manner as to support the sill directly under the posts of the upper set. Sometimes, instead of diagonal braces, heavy caps are placed on the posts, across which are special stringers to pass under the posts of the upper set. If the timbers of the upper set are tightly compressed, a whole set may be exposed without fear of collapsing.

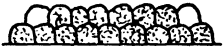


FIG. 51. CROSS-CUT METHOD

With the cross-cut method (Fig. 51) the bridge is taken out in horizontal strips. A cross-cut about 7 ft high is started from a winze and driven to the limit. It is then filled as tightly as possible against the back, preferably with large pieces as they shrink less than fine. If there are not sufficient large pieces even to build a wall against the ore, timber must be used to keep the fines in place. The cross-cut is timbered with clap-me-down sets, the legs of which are tapered and rest on sills. About 6 in. from the top an inch hole is drilled through into which a drill can be inserted to help pull the leg out when cross-cutting the next higher horizontal strip. When the first cross-cut is filled, another cross-cut is driven alongside the first in a similar manner, and so on, till the whole slice is worked out. Each successive slice is worked in the same manner. The first cross-cut, though undermined, costs more than the other cross-cuts on the same level, as, in addition, the latter have a free side.

In some cases a drift is driven in the ore and stoping commences above the roof of the drift. This leaves a horizontal pillar of ore level with the drift and this helps to take much of the weight off the drift timber. When coming up from below, the drift timber is secured in the usual way, and the horizontal pillar extracted by the cross-cut method.

Top Slice Caving. This method is confined to wide, weak deposits near the surface which will only stand unsupported for a very short time, and is overlaid by a friable overburden that will break up and cave promptly. It can be used for thick tabular deposits lying at any angle from horizontal to vertical. The levels are immediately below one another. Practically all the ore is extracted with little danger of falls. A considerable amount of timber is required and lost, and there is much handling of both ore and timber.

Sub-level Caving. This method can be used in cases where top slicing would be dangerous because of the hanging up of the gob. The body of ore is developed by a series of sub-levels from 18 to 50 ft apart, but they need not all be driven before commencing to cave the upper sub-level. This method is used for mining smaller deposits than would be suitable for block caving, but it is also applicable to large ore bodies, and to soft ores which will stand fairly well for a brief time over narrow spaces, but will cave over wider openings.

Block Caving. The deposit should be wide, uniform, and readily broken up, with an overburden of sufficient depth to give weight and crush the ore body after it has been undercut. Applicable to large low-grade deposits only, where the loss of ore, or its dilution with waste, is permissible. The ore should be free running and must not pack. The blocks are 200 to 250 ft long, about 100 ft high, and the width of the deposit. A large amount of development is not necessary. The caving is induced by freeing the ends through underhand stoping 8 ft wide. The bottom of the block is cut up by drifts and cross-cuts, the pillars are then reduced as much as is consistent with safety, after which they are blasted. The old method of extraction by caving to the main level has given way to a number of chutes below the block which extend to a haulage drift some distance below, with an intermediate grizzly level where large pieces are broken up: this virtually does away with hand shovelling. Danger from falls is slight. Development and mining costs are low, and a large output possible.

Mining of Coal

Coal. One cubic yard of coal in the solid is generally taken as a ton. To ascertain the tonnage of coal per acre, multiply 62·255 (the weight in lb of 1 cu ft water) by the sp. gr. of the coal (1·14 to 1·40) which gives the weight per cubic foot. This result multiplied by 43,560 (the number of square feet per acre) and the result multiplied by the thickness of the seam in feet, gives the number of pounds of coal in one acre. If this is divided by 2240 (the number of pounds in a long ton), the quotient expresses the number of long tons per acre. Green (freshly mined) coal clings to clay bands for about an inch on each side, but when weathered it is easily detached. It is, however, not worth the trouble of collecting so must be looked on as so much lost. Round coal is that over ¾ in. diameter; slack is that under ¾ in., while duff is coal dust. Lump coal is preferred for shipping as it stands a certain amount of handling, and though some of it is broken up, a fair amount of round coal can be delivered to consumers.

Most coal seams have two systems of cleavages known as "cleats" or "facings" which cross each other at approximately right angles. The main or face cleats are more prominent than end or butt cleats. The direction in which the coal is attacked with respect to the cleats is of prime importance, since it greatly facilitates or retards the mining of coal. The direction in which the face is worked is known as "end-on" when the breast is driven at right angles to the main cleat. This method is used when roof pressure is very heavy, when the coal is very soft, and the cleats are working free. When the breast is driven parallel to the main cleats it is called "face-on" direction; this is adopted when the main cleats are not so free or numerous as the butt cleats. "Half-on," "longhorn," and "shorthorn" are various directions between end-on and face-on. The direction in which one works depends not only on the cleats of the coal and joints of the roof rock, but also on the dip of the seam, for a suitable grade for the roadways must be secured. It is better to work the face to the rise than to the dip as there is less tendency for the roof pressure to crush the coal at the face, and there is less difficulty in haulage and drainage.

The conditions that formed one seam are generally repeated in the same basin, but all the seams are not of the same quality. For commercial purposes one naturally works the best seam first in order to secure a market and obtain the best price: but in doing this other seams in the same basin may be ruined. If the inferior seam is overhead, it may be so broken up by the extraction of the lower seam as to be too expensive to mine, especially when the lower price it will fetch is considered. If the inferior seam is below the one that is worked, there may be difficulty in supporting the roof. If circumstances permit, inferior coal may be used at the mine in gas producers and used for generating electricity, thereby saving its loss.

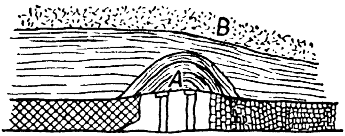


FIG. 52. LONGWALL MINING
A, Underweight; *B*, Overweight of roof

Dimensions in collieries are generally given in yards instead of feet as in metal mines.

There are two main methods of mining coal: the longwall and the bord-and-pillar, with their various modifications.

The conditions which are suitable for longwall are—

(*a*) Narrow seams, between 3 and 6 ft thick, of regular thickness.

(*b*) A moderate pitch, between 0 and 20°.

(*c*) Plenty of waste to fill the gob or goaf.

(*d*) Regular and steady work throughout the year.

(*e*) When the surface may be let down without much injury to it.

(*f*) Hard coal capable of bearing pressure and which parts freely from the roof.

(*g*) Deep mining.

(*h*) Seams liable to spontaneous combustion (longwall retreating).

(*i*) There must be freedom from "hogbacks" (rolls in the floor, but not in the roof), "horsebacks" (a washaway in the coal filled with shale or sandstone), undulations and faults.

The fractured roof near the face forms an arch known as the "underweight." This rolls forward, pressing on the rib of coal undercut and breaks it down when the sprags are removed. The "overweight" is the solid strata which settle down on the packs in the worked out ground. If the regularity of work is stopped by strikes or any other cause, the underweight presses down on unholed coal making it more difficult to work when once the travelling weight stops, and it takes some time before it can be started again. Flat quartz veins cannot be worked satisfactorily on the longwall principle on account of the difference in the relative rates at which the faces of coal and quartz are worked. In coal mining the roof curves down gradually; in vein mining the roof forms cracks more or less parallel to the face which breaks up the hanging wall and makes it very heavy on the supports.

There are two kinds of longwall mining: longwall advancing and longwall retreating. The former is generally carried out as coal is won almost from the first and is gradually increased. With longwall retreating, the mine has to be opened up first which means a large capital outlay. The better plan, when it can be carried out, is to work half the mine advancing, and the other half retreating. Longwall retreating is, in principle, preferable to longwall advancing for the thicker seams, for those liable to gob fires, and which have a treacherous roof. It is easier to maintain haulage roads in coal than in the gob. The seam is proved before 10 per cent of it is extracted. All airways are tight and the ventilation is almost perfect. It is comparatively easy to seal off a fire. Pairs of cross entries (headings or gangways) are driven to the required distance and the block of coal between each pair is then worked home, i.e. towards the shaft. The face of coal may be straight or semicircular. If straight it is sometimes expensive to maintain. When this is so, the stalls or working places may lead one another a short distance, forming horizontal steps or minor curves, their size and direction largely depending on the freeness of the cleat. The direction of the face depends chiefly on the dip of the seam and the direction of the main cleats. The passage through the gob is known as the "mother gate," and branches from it are called "cross-headings," branches from the latter are known as "stall roads." The branches are generally oblique to the road from which they branch: by this means the length of the gob roading to be maintained in repair is kept within reasonable bounds, the distance the coal has to be hauled is reduced, and the skips or cars are easily turned out from one roadway to another.

Advantages of the longwall methods as compared with bord-and-pillar are—

1. The small amount of narrow work (six yards wide or less being considered narrow work and paid for at a higher

rate than wider work on account of more frequent tight corners).

2. The mine becomes remunerative sooner.
3. The yield per acre is greater, as less coal is lost in pillars and ribs.
4. Less small coal is made as when driving several narrow places and by the cracking of the edges of pillars.
5. The men are more concentrated and therefore better supervised.
6. The ventilation is easier.
7. There are less roads to keep up.
8. Shot firing can almost entirely be done away with, which is specially advantageous in a fiery mine.

The conditions suitable for bord-and-pillar mining are—

(*a*) Thick seams.
(*b*) Roof that stands well.
(*c*) Absence of gobbing material.
(*d*) The surface must be left intact.
(*e*) Soft coal.
(*f*) Seams cut up by faults, rolls, horsebacks, clay bands, or other irregularities can be worked.

The objections are the cost of driving so many narrow places which also increases the production of slack; the amount of coal lost in pillars and ribs. In the best modern practice never more than 30 to 35 per cent of the coal is removed in the "whole" (i.e. working out the headings and bords): if more than 40 per cent is taken out, there is great danger of creep, while if 50 per cent is removed a creep is almost inevitable. Bords are driven off the haulage on one or both sides. So as not to weaken the pillars bounding the entries, the neck of the bord is made narrow and widened inside to its full width. Pillars should be worked as soon as possible after the bords have been driven, if not required to support the surface, because this saves coal from being crushed, saves maintenance, improves ventilation and is safer. The pillars must be worked systematically and in step form so as not to make a line of weakness. Bords should be turned off from the intake, not from the exhaust heading. Narrow bords may be driven in horizontal steps, and as soon as one is finished it is followed by pillar drawing. By this method most of the coal is won and there is little danger to the miners.

With the panel system, the colliery is divided into a number of smaller mines of about 30 to 40 acres which are surrounded by barrier pillars 40 to 60 yards wide which are only holed

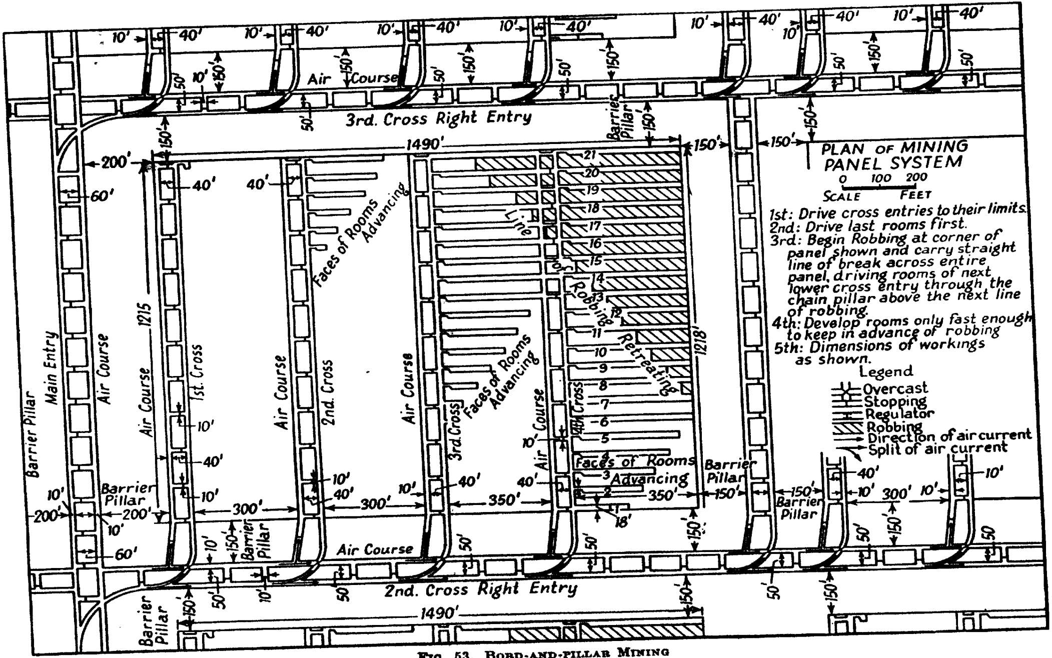

PLAN OF MINING PANEL SYSTEM
0 100 200
SCALE FEET
1st: Drive cross entries to their limits.
2nd: Drive last rooms first.
3rd: Begin Robbing at corner of panel shown and carry straight line of break across entire panel, driving rooms of next lower cross entry through the chain pillar above the next line of robbing.
4th: Develop rooms only fast enough to keep in advance of robbing
5th: Dimensions of workings as shown.
Legend
Overcast
Stopping
Regulator
Robbing
Direction of air current
Split of air current
Barrier Pillar
Main Entry
Air Course
3rd. Cross Right Entry
2nd. Cross Right Entry
1st. Cross
2nd. Cross
3rd. Cross
4th Cross
Faces of Rooms Advancing
Line of Robbing Retreating
1215
1490'
1218'
200'
150'
300'
350'
60'
50'
40'
10'
18'
1
2
3
4
5
6
7
8
9
10
11
12
13
14
15
16
17
18
19
20
21

FIG. 53. BORD-AND-PILLAR MINING

through where necessary for roadways. Any method of winning the coal can be used within a panel.

The advantages of the panel system are—

(*a*) Better control of ventilation.

(*b*) The risk of explosion is decreased as each district has its own current of air.

(*c*) Each panel when worked out can be sealed off.

(*d*) The roof does not weigh so heavily on soft coal.

(*e*) More lump coal is produced.

(*f*) In case of fire within a panel, it can be sealed off.

Holing by hand is very trying to miners owing to the constricted position in which they have to work. Coal-cutting machinery has been designed for undercutting and shearing. They not only relieve the collier of the heavier part of his work, but decrease the cost of mining. There are various types of coal cutters for use under different conditions. They are worked by compressed air or electricity. If a mine is gassy, and electricity is employed as the motive power, special precautions must be taken to enclose those parts liable to spark. Some of these machines are self-propelling, while others have to be moved about on loaders or trucks. Coal-cutters may be constructed so that they can work at any reasonable height from the floor.

The mechanical-pick or puncher is somewhat like a rock drill inasmuch as it moves forward and backward, but it does not turn. It can be used in any place accessible to hand-pick work: where the face has to be supported within three or four feet of the face and there is no room for a chain-breast machine which requires a clear space of ten to twelve feet: also where brasses or other hard materials occur which might break the bits of a chain machine, the puncher can work round such obstruction. The puncher makes cuts 8 to 10 in. high in front, to 2 in. in the rear, so that when the coal drops it shall fall forward. The machine can be made to exhaust on either side so as not to annoy the helper when shovelling the slack; but the exhaust is cold for the feet of the runner. The puncher is not suitable for dips over 12 to 15° on account of the difficulty in moving the machine from one cut to the next. If run fast, say over 190 blows a minute, a large proportion of them are struck at random. The number of blows are independent of their force. The machine, which rests on a four feet inclined platform to keep it up to its work, is moved from side to side, and when one cut is completed, the machine is moved on to another platform alongside for the next cut.

The chain-breast machine is used in entries and bords in places where the coal seam is over 30 in. thick, and at an angle not more than 10 to 12° from the horizontal. It works fast, makes very little slack in the cutting, and is not subject to the racking action of the punching machine. There is an outside frame which is moved sideways, and an inner or sliding chain-cutter frame which moves forward. On this frame is placed an endless chain which is recessed to receive a bit or cutter. The movable frame carrying the chain is fed forward at a speed determined by the resistance offered by the coal. The machine is held in place by a rear jack set in the roof at an angle. Much of the working time is occupied by withdrawing the machine from the cut, resetting the jack and moving the machine into its next position. The undercut being of the same height all the way to the back, the block of coal is liable to drop down rather than fall forward. A chain-breast machine makes about 60 per cent of the slack a puncher does; but this is not always an advantage as when a low horizontal undercut is made the coal does not fall forward, so often the front of the cut has to be enlarged. Nodules or balls of pyrites may sometimes be removed by reversing the machine to clean out the cut, removing some of the cutters that come in contact with the obstruction and then starting the machine again. The cutters that are left in cut over and under the balls, and then the cutters which are in line, suddenly take a deep hold and may tear out the nodule. Chain machines are seldom used for working pillars, as the weight of coal bears down on the machine which may cause damage or necessitate them being cut out with hand picks. It is for a similar reason that chain-breast machines are unsuitable for longwall work, and because too much space is required for the machine between the gob and the face of the coal; besides it takes too long to move a chain-breast machine for longwall work where a machine is required that will cut continuously along a face to keep the underweight moving. In most types of chain-breast machines it is possible that the cuts may be put in at different heights, making an uneven floor or leaving a rib between cuts. The machine being rigidly fixed is unable to follow closely irregularities in the bottom; also the rear jack piercing the roof at regular intervals may bring the roof down, particularly since the length of the machine makes close propping difficult.

The shortwall machine is very similar to the longwall machine of the same type. When once the first or tight cut is made, the arm, instead of working at right angles to the machine, is worked end on. When the arm has cut its way into the coal, the whole machine travels across the room or bord, not stopping till the breast is completely undermined. The speed of the machine is from 14 to 18 in./min.

There are two types of longwall machines. One is a chain machine, the other a rotary disc. They are of necessity but a few inches high. The cutting part of the former is an endless chain containing the bits which operates along a narrow frame or arm which when at work extends from the side of the machine. The swing of the cutting frame allows a cut to be made as far as desired, short of the full extent of the arm. The arm can be worked from either side. Longwall machines are not suitable for very friable coal, since that can be holed cheaper by hand. The operator requires two helpers; a boy and a man. The boy shovels the slack at the side, and the man behind throws it under the face to hold up the coal instead of sprags. The chain pulled on by the operator to move the machine farther on is held to the floor by a bar split at the bottom and braced against the roof.

The rotary or disc coal cutter is practically a horizontal circular saw. The machine runs on ordinary rails and no fixing is required to keep it at its work. The operator regulates the speed of feeding, the cable being wound on a drum at the rate of 7 to 25 in./min, the other end of the rope being fixed. A disc cutter 4 ft in diameter will make a cut 3 ft 6 in. deep. The angle of the cutter wheel can be adjusted with respect to the horizontal so that it can follow the formation of the floor and avoid impurities. The kerf or height of the cut is from three to four inches.

Passes

Passes may be at an angle or vertical. Most ores will run well on an angle of 60 to 65°. For sticky ores a pass should be inclined as it is liable to pack in a vertical pass. To prevent packing a pass may be at an angle for half its depth and then knuckle back at an angle in the opposite direction for the rest of the way. Inclined passes wear out more quickly than vertical. If an ore pass is run empty, a big chunk falling down might break the chute gate at the bottom. Passes may be made single or double. When double one compartment can be used as a ladderway, but must be protected on the top from ore being shovelled into it by mistake which might cause serious injury to anyone travelling up or down. When the ore compartment shows signs of wearing out, it can change over to the ladderway. An ore pass is generally built up as the stope nears the level above. Passes are generally rectangular in shape and constructed of logs. As a rule the life of a pass is that of 100 ft stope. Passes are sometimes made circular so as to avoid dead corners: these may be constructed of wood, stone, or iron. The distance apart of passes largely depends on the comparative cost of building the passes and the method of mining; the rill method not requiring so many as the flatback for drawing the ore, unless a large output is required, when there must be several passes from which the ore can be drawn. The distance apart of passes should not be farther than a man can shovel when working between them unless barrows or small stope cars running on temporary rails are used. The maximum position of the lip of the chute at the bottom of a pass is the vertical line above the near rail, and this should not be less than 4 in. nor more than 6 in. above the side of the car to be filled. There are various kinds of gates used for chutes depending on whether the ore is coarse, fine or mixed.

To prevent a pass from getting hung up, certain precautions can be taken, such as placing bars across the mouth of the pass to prevent pieces larger than ten inches entering; being careful to see that drills do not find their way into the pass where they may get crosswise and hold up ore. Ore should not remain too long in a pass; it must be kept moving. If the ore is clayey, the pass should be run empty. The method of freeing a hung-up pass depends on the nature and position of the block. If near the chute it may be barred down. If the block is chronic owing to faulty construction, or the nature of the ore, a chain fastened inside at the top of the chute when shaken may start the ore running again. Sometimes the ore may be started if the block is higher up by tapping the timber from the man-way, or a hole may be cut through for a crowbar to pass through. When the block is part way up the pass, a plug of dynamite may be fastened to one end of an iron rod, or a series of iron piping. In the former case the fuse is tied to the rod, in the latter the fuse passes inside the pipes. This is pushed under the block and fired. Another method is to use a so-called cannon, which is made out of a piece of shafting 12 in. long and 4 to 5 in. diameter. A 2 in. hole is bored into this for 10 in., and the cannon further strengthened with three wrought-iron rings ½ in. thick. A touch hole is bored ½ in. from the bottom of the hollow for the fuse. The full charge for this cannon is 10 to 14 oz blasting powder but sometimes less is used. Dynamite would be too strong for the cannon. A lump of rock is often placed on the mouth of the cannon; but it is the shock rather than the lump of rock that loosens the block. The cannon is placed at the bottom of the pass with the mouth pointing upward and has a chain attached to it so that it can be easily recovered. If these means do not avail or the block is near the top of the pass, the material must be dug out by a man with a rope attached to him in case the ground beneath gives way.

Coal is soft and gets more or less broken up every time it is handled, therefore it is sent to the surface in the cars in which it was originally loaded.

Chute Gates

A gate should be designed so that a car can be quickly filled and the gate closed with little spillage before the car overflows.

The Chinaman or Platform Gate. This is mostly used in narrow veins where there is no room for any other kind of gate. It consists of short planks placed across stringers resting on caps in a drift. Ore rills on to the planks which are moved one at a time as required to allow ore to fall into the car below. The planks are then replaced.

Scraper Loading Gate. This is somewhat similar to the above inasmuch as it delivers ore to a car from overhead, but the hole through which it falls is permanently open and remains in the same place all the time.

Drills. If ore is very coarse it may be held back by means of two or three drills passed through holes made in the sides of the chutes.

Projecting Boards. Two or three boards supported on iron-bar brackets fastened to posts on either side of the chute.

Stop Boards. There may be two or three boards which slide in grooves inside the chute. These may be vertical or inclined towards the inside of the chute. Only one board reaches the bottom of the chute, the others stop at different heights like steps. With these two last types the trucker is liable to get his fingers hurt.

Vertical Guillotine. This is a common type consisting of a piece of boiler plate worked by a hand lever and link from one side. If a piece of ore gets jammed between the bottom edge of the gate and the floor of the chute, fine material, if present, will escape into the drift.

Arc Gates. These are used to obviate the spillage as in the case of the Guillotine type. There are two kinds; the overcut and the undercut. The undercut cuts upward through the stream of ore and effectually stops the flow of fines. Large chunks are either tumbled back into the chute or forward into the car.

Shark's Mouth Gate. This consists of two jaws of wood 9 by 2 in. The outside one is bolted on to the inside jaw, not far from the handle, the inside jaw is bolted at its far end to a post. It is kept in place by long iron dogs. The handle of the smaller jaw is bolted on to it in such a way that it comes outside the dog on the side from which it is worked. This gives finger room when operating it. To fill a car, both jaws are raised together, but when nearly full the larger jaw is lowered to cut off the bulk of the ore; the smaller jaw regulates the final filling of the car. This gate is suitable for quick loading of a car with ordinary run-of-mine ore, but is not suitable for very fine ore. It can be quickly unbolted and removed to another chute.

Swing Gate. When lowered, this spans the space between the chute and car or skip. When not in use it is drawn up out of the way and prevents ore from flowing out of the chute or bin.

Finger Gate. Instead of a solid gate, five wooden fingers hang down in front of the opening of the chute, each being independent of the other. This is used for coarse ore. Its chief advantage is that large pieces can be removed, one at a time, by lifting one or more fingers separately. They are relatively high in cost, require considerable room, and are cumbersome.

Chain Gates. Heavy iron chains hang down in front of the chute. They can be raised by hand, rope, or with an air piston. They last a long time, do not take up much room, and are not injured by large pieces falling against them, or by blasting if a chute becomes blocked.

Handling Loose Material Underground

Provision has to be made to remove broken ore or coal from the face. The size of a hand shovel used should be in proportion to the weight of the material to be shovelled, and the strength of the shoveller. If shovelling on a smooth surface a square-nose shovel can be used, but otherwise a round-nose shovel should be employed.

A miner's wheelbarrow has no legs. It is therefore not easily upset; also the man filling it does not have to lift the material so high as if it had legs.

Coal will not move on an angle up to 25° from the horizontal, unless the floor is covered with sheet iron. Coal will slide down a smooth surface between 25 and 30° if not damp and it does not contain too much fines. For angles up to 30° breasts are worked empty. For angles over 30°, gates are required at the bottom to prevent the coal from rushing down. For narrow nearly horizontal beds or veins, shaking troughs may be suspended from the roof or a framework by chains. In longwall mining a stationary square-bottom, wrought-iron trough, which can be easily moved up to the face from time to time, has a chain circulating in it which moves the coal forward. Such conveyers have reduced the number of gateways formerly used.

Slushers of various kinds are used to scrape up loose material. No one machine suits all conditions. They can be used in certain stopes as well as on the ground floor. Some have no bottom. The box type of scraper has bottom, sides and back and is used for fine material; this type is liable to ride over lumps. The hoe type has little or no sides and is used for handling coarse material. Another type has teeth on one edge and is smooth on the other. The teeth comb lumps off a pile; the scoop is then turned over and the smooth edge used for scraping the finer material. The digging angle found most satisfactory in practice for average conditions is about 20°. The scraper is worked by a two-drum hoist and wire ropes. The material can be scraped up a ramp so that it can fall into a car below. Blocks are used for the tail rope to run through. On short hauls (under 75 ft) a speed of 150 ft/min is sufficient. When picking up its load, the scraper must move more slowly. If the bottom on which the scraper is dragged is rough, the load will be spilt if worked at a high speed. If the track is smooth, the speed can be increased. The weak point in scraper loading is the wire rope which must be small and flexible, necessitating small wires, which, rubbing on a rough bottom, shortens its life.

Where there is sufficient room, broken material can be scooped up by a mechanical shovel and loaded into cars.

Belt conveyors are sometimes used underground for conveying broken material. They may be employed for conveying coal or ore from the face, or for delivering filling to depleted stopes. Belt haulage must be in a straight line, and the feed should be in the direction of the moving belt. Belts for a given capacity, if of insufficient width, wear out more quickly than if wider as they have to travel faster and pass more often under the point of loading, where most wear takes place. A belt should have the proper number of plys for the nature of the work it has to do. For coal, the rubber cover should be

$\frac{1}{16}$ in. on the top surface; while for lump ore it should be $\frac{3}{16}$ in. on the top and $\frac{1}{32}$ in. on the bottom. The speed of a belt conveyer varies with the nature of the load. For run-of-mine coal it may be 300 ft/min, while for coarse stone it may be only 200 to 250 ft/min. The tension of the belt is important. The working tension per inch of belt width per ply of 20 oz is 24 lb; for 32 oz duck is 27 lb; and for 36 duck is 30 lb. For belts over 24 in. in width it is advisable to use 5-pulley type idlers as being less likely to form sharp bends than the 3-pulley type. The normal spacing for idlers to carry the belt is from 2 ft 6 in. to 5 ft: they are generally closer together at the feeding point. Return idlers can be spaced 10 ft apart.

Testing Safety Catches

Where hoisting is not so fast as to cause safety catches to act when they should not, safety catches are generally attached to cages. When this is so, they should be tested periodically. This should be done from the drum, not from the shackle, as in the latter case the back drag of the rope, should it break or come off the drum, might prevent the catches from acting at once. The mouth of the shaft is covered over with strong timbers on the top of which are placed sacks of some soft material such as chaff to take up the shock in case the grips do not act. The cage is raised a few feet above the collar of the shaft. A ring bolt is let into the ground near the drum, to which a short piece of rope is fastened. Another short piece is clamped to the hoisting rope nearby. The two short pieces are now connected by a slip-hook. The winding rope is slacked off the drum so that the subsidiary ropes take the weight. By opening the slip-hook, the cage is given an opportunity to fall. The catches should act within one or two inches.

Tests of Materials

It is as well to know certain rough and ready tests that can be made at a mine where proper appliances are not available, both for impurities and strength.

Cotton Waste. Test the absorbent properties compares with known good grade cotton waste. To ascertain the amount of inorganic matter present (for it is sometimes adulterated with barium sulphate to give it weight, though it still looks white and clean), burn a given weight to ashes and then weigh the amount of inorganic matter left.

Cord Wood. When firewood is sold by measurement instead of by weight, specify that, if the timber is hollow, and the pipe is more than 3 in. in diameter, it must be split so as to make it stack closer. See that the contractor does not plug up the ends of pipy logs with pieces of saplings to make them look solid.

Manilla Rope. A good manilla rope is hard, pliant, yellowish or greenish grey, with a certain silvery or pearly lustre. When making an examination, cut a section of the rope to ascertain if inferior kinds of hemp have been used in the centre. If of a dark or blackish colour it indicates the rope has suffered fermentation in the process of curing; brown spots show that the rope was spun while the fibres were damp, and consequently softer and weaker in those parts. Inferior ropes are made of short fibres or with strands of unequal thickness, or are unevenly spun. In the first case the number of short ends make the rope appear woolly; in the second and third cases the irregularity is noticed on inspection. The breaking strain of a new hemp rope is one ton for each pound weight per fathom. A few months of exposed work weakens a rope considerably. A rope wound round a barrel offers the following resistance to slipping: 24 when the rope is wound once round the barrel; 111 when wound round 1½ times; 535 when wound twice and 2575 when wound 2½ times. An eye splice over a thimble has 90 per cent efficiency of the rope: a short splice has 80 per cent: a timber hitch 65 per cent: a bowline or clove hitch 60 per cent: a reef knot or sheet bend 50 per cent.

Steel Ropes. The following are the breaking strains of different kinds of steels per square inch. Mild steel, 40 to 50 tons. Best crucible steel, 50 to 60 tons. Best patent crucible steel, 70 to 80 tons. Best plough steel, 110 to 120 tons. Mining ropes are generally made of the best crucible steel. Steel ropes are subject to corrosion as well as to abrasion and fatigue.

To test a wire rope cut off a length from the shackle end equal to at least a quarter of the circumference of the pithead pulley, as that is where the strands are liable to open and allow water to reach the core, so it is well to change its position. The shackle end is also liable to sudden jerks when winding is started from the bottom of a shaft. Clean the sample piece: untwist it to examine the core and note if it is destroyed or the inner wires corroded. The tensile strength of each wire is tried, the results added together and diminished by 33 per cent, as that is the loss in strength due to twisting. The flexibility is tested by placing each wire in a vice with jaws rounded to a radius of $\frac{3}{16}$ in. and noting the number of times it can be bent from side to side at an angle of 180° without breaking. The wires of a new rope will stand from 14 to 17 flexions. When found to break with five flexions, the rope should be changed. The number of twists that can be made in, say, a length of 8 in., should also be noted. A new rope should be tested twice a year: as it gets older, it should be tested more frequently. No standard rule can be given as to how long a rope will last as the conditions under which they are used vary so much. The usual factor of safety allowed for a hoisting rope is $\frac{1}{6}$ to $\frac{1}{7}$, the breaking strength of the rope. A hoisting rope should be examined at least once a month by passing it through a gloved hand or a handful of cotton waste, when any broken wires can be felt, or a few threads of cotton waste are caught in them. The rope should not move at a rate of more than 40 to 50 ft/min. If a broken wire is long, it is tucked in among the others; if short, it is broken off as the fracture will be nearer to the rope than if it were cut. If there are six broken wires in one rope lay, the rope should be discarded. If necessary to splice a wire rope, the splice should be from 30 to 50 ft long.

Blasting Powder. A good blasting powder has a uniform dark grey or slaty colour before being coated with graphite, and should not show any difference in colour when crushed. If bluish or quite black, it indicates the presence of too much charcoal, or it is damp. A diversity of colours shows that the ingredients were unequally mixed. Light coloured spots show that saltpetre has effloresced during drying, thus disturbing the uniformity of the mixture, and reducing the strength and reliability of the powder. When poured over a sheet of paper, no dust should be left behind, for such dust indicates either the presence of moisture, or of fine mealy powder, whereas the size of the grains should be uniform. On pressing between the fingers, a crackling sound due to sharp cornered grains should not be heard, nor should the grains be easily crushed. A pinch of good powder ignited on a sheet of white paper burns rapidly away, leaving no residue. If black spots remain on the paper, they show an excess of charcoal or a poor mixing of the ingredients. Yellow spots indicate an excess of sulphur. If holes are burnt in the paper they indicate an excess of moisture or some other imperfection. If the powder is slightly damp and the saltpetre has not separated out, it may be dried in the sun without deterioration.

Dynamite. If the paper case of a nitroglycerine compound has a greasy feel, it shows there is a leakage of nitroglycerine. A white efflorescence caused by the separation out of nitrate of potash shows that the "dope" or absorbent originally contained an excess of moisture, or that the dynamite had been stored in a damp place: anyhow, nitroglycerine is sure to leak out of such damaged dynamite, and being dangerous in consequence should be destroyed. Nitroglycerine when decomposing generally gives off nitrous fumes which impart a green stain to the inside of the cartridge. The presence of free acid can be determined by blue litmus paper. In order to ascertain if dynamite is leaking, place it on a clean sheet of brown paper in a room heated from 15½ to 21° C. Good dynamite should show no discoloration of the paper.

Safety or Time Fuse. This should burn at the rate of 2 to 3 ft/min. When a fuse begins to crackle, it is a sure sign that it is running. A piece of good fuse pressed in a vice fitted with wooden jaws ought to burn while under compression up to 300 lb. This test will show whether the fuse will stand up to tamping. The water resisting power of a fuse may be tested by immersing a piece 4 yd long, with the exception of 6 in. at either end, in a vessel of water and subjecting it to a pressure of 10 lb/sq in. for 20 min, after which one end of the fuse is lit, when the fire should travel the whole length of the fuse. To test for uniformity of burning, take two pieces of 2 yd each, place them together and ignite both at the same time: the difference in the time of burning should not exceed 5 per cent.

Sand for Concrete. Organic matter, whether animal or vegetable, is objectionable in concrete. Place some of the sand in a white glass bottle with a 3 per cent solution of caustic soda and allow the mixture to stand for 24 hours. The bulk of the solution should be equal to that of the sand. If the colour of the solution is not more than a pale yellow, it may be considered free from objectionable organic matter; but if of a decided yellow or brown colour, the presence of organic matter is indicated and the sand will not produce a good quality concrete.

Lubricating Oil. High-grade lubricating oil is sometimes adulterated with low-flash mineral oil which can be determined by testing the flash point. The presence of vegetable or animal oil in mineral oils is determined by mixing a sample with a solution of caustic soda or caustic potash and heating to about 200° F. Vegetable and animal oils will saponify and separate from the mineral oil. Acids are determined with litmus paper.

VENTILATION

Pure air is composed of 20·93 per cent oxygen, 79·04 per cent nitrogen, and 0·03 per cent carbon dioxide. An increase or decrease to any extent of these gases upsets the balance, and is harmful to man. In regulating the amount of air in a mine per man, it must be remembered that in a confined space man cannot consume all the oxygen present; also that there are other oxygen-consuming substances to be provided for. The range between the maximum oxygen content of normal air and the minimum safe content is extremely narrow, approximately 5 per cent. The number of respirations made by a healthy man between the ages of 25 and 30 years is 16 per minute, and he takes in 34 cu in. with each respiration. If through the oxidization of minerals underground, the amount of oxygen in the air decreases, the proportion of nitrogen increases; and since nitrogen does not support combustion, anyone breathing such an atmosphere becomes suffocated.

Gases Encountered

Carbon dioxide, the black damp or choke damp of coal miners mixed with nitrogen is not only given off by the breath of man and beast, but also by certain rocks such as limestone and andesite. Being heavy, it sinks to the ground. When descending old shafts, one should always test the air by holding a *naked* light low down. If an excess of carbon dioxide is present, the light will first flicker and keep alight with difficulty and finally go out, which serves as a warning for one to retire.

But in mines there are other gases encountered, such as methane, carbon monoxide, sulphuretted hydrogen, and nitric peroxide.

Methane is given off by certain coals: it will not support combustion but becomes inflammable when 5·55 per cent is present in the air, and explodes with 7·14 per cent which increases in violence up to 9·46 per cent, after which the violence decreases and ceases altogether at 16·67 per cent but still remains inflammable till the mixture contains 29·5 per cent when it extinguishes a flame. In coal mines the gas is not pure and is known as "fire-damp." The test for fire-damp should be made every morning in a fiery mine, by an experienced man, before the miners go to work. Methane being lighter than air ascends to the roof, and is tested with a hydrogen flame as being more delicate than oil. It will indicate from ¼ to 3 per cent.

Carbon monoxide, or white damp of coal miners, is caused by incomplete combustion due to explosions and fires in places (mostly in collieries) where there is insufficient air. This gas is also lighter than air. It is an accumulative poison. It used to be tested by taking white mice or canaries in cages to the testing place, and holding them high up; these creatures are preferred as being tame and light in colour and therefore easily observed. Nowadays better methods are employed. Hoolamite, or activated iodine pentoxide, is used as an indicator, which can detect as little as 0·07 per cent, a determination being made in less than a minute, whereas it takes about 30 min to obtain indications from a mouse or a small bird. Palladium chloride paper can also be used: the time taken to darken the paper depends on the percentage of carbon monoxide present. With 0·01 per cent the darkening of the paper begins in 11 min and is complete in 60 min; 0·1 per cent takes 1 sec and 9 min; 1 per cent takes 16 sec and 2 min. A qualitative detection of carbon monoxide in air in amounts greater than 0·03 to 0·04 per cent can be made by shaking a dilute solution of blood with the air and then comparing it with an unexposed sample of a similarly diluted blood. If carbon monoxide is present, the exposed sample will be pinker than normal blood.

Sulphuretted hydrogen in a diluted state is recognized by its characteristic odour of rotten eggs. It is poisonous, and even as little as 0·01 per cent may cause death. It is occasionally met with when breaking into old workings. It is inflammable, and should a naked light be near when this gas is tapped, it will catch alight and surprise the unsuspecting miner and possibly singe his hair.

Nitric peroxide is formed by the burning instead of complete combustion of nitro-glycerine compounds largely due to using detonators too weak to explode the dynamite properly. The breathing of nitric peroxide gives symptoms similar to those of pneumonia, but may not show themselves till several hours after inhaling.

Gassing

The specific gravity of pure air is taken as 1. Oxygen has a sp. gr. 1·1053. Mine air may be satisfactory with regard to the standards of oxygen, and yet be unsatisfactory with regard to impurities. In regulating the amount of air per man in a mine, it must be remembered that man cannot consume all the oxygen in the air; also that other oxygen-consuming substances must be provided for. It is generally considered that one man underground requires about 20 cubic feet of air a minute, but it is usual to provide him with 50, 70, or even 100 cubic feet per minute, and in heavily-timbered mines it may have to be increased to 200 cubic feet. The range between the maximum contents of oxygen of normal air and the minimum content is extremely narrow; approximately 5 per cent.

Nitrogen has a sp. gr. 1·9673. When oxygen of the air is consumed by oxidation of sulphides nitrogen is left behind. This does not support life; consequently anyone breathing it is suffocated.

Carbon monoxide, sp. gr. 0·9670, is lighter than air, and consequently rises to the top of a passage. It is formed by the incomplete combustion of carbonaceous matter. It is an accumulative poison. The danger of carbon monoxide is that it displaces oxygen from its combination with haemoglobin of the blood. The former method of testing the presence of carbon monoxide with small warm-bodied animals is too slow. It is more quickly determined qualitatively in amounts greater than 0·03 or 0·04 per cent by shaking a dilute solution of blood with the air, and then comparing it with an unexposed sample of similarly diluted blood. If carbon monoxide is present, the exposed sample will be pinker than normal blood. Hoolamite or activated iodine pentoxide is used as an indicator for carbon monoxide and can detect as little as 0·07 per cent present. A determination can be made in less than a minute. Palladium chloride paper can also be used for determining carbon monoxide in air. The time taken to darken the damp paper depends on the percentage of monoxide present. With 0·01 the darkening of the paper begins in 11 min and is completed in 60 min. 0·1 per cent takes 1 min and 1 per cent 16 seconds.

Carbon dioxide, sp. gr. 1·5288, is not poisonous in itself, but by replacing oxygen in the air causes suffocation. Being heavier than air, it collects in low places such as shafts and winzes, and must be removed by stirring it up before a man can enter such places. Its presence is indicated by lowering a naked light into a suspected place. At first the flame is difficult to keep alight, and eventually is extinguished. Carbon dioxide is given off by limestone and certain igneous rocks, by rotting timber, combustion of naked lights, and by the breath of animals.

Methane, sp. gr. 0·5540. Mine gases are never pure, they are always mixtures; but one gas predominates over the others. The coal miner's firedamp is mostly methane. It may be held in seams mechanically as occluded gas, or chemically in bituminous matter. Being lighter than air, it tends to rise to the top of an enclosed excavation. Its presence is determined by holding a hydrogen flame in a safety lamp overhead.

Nitric oxide, sp. gr. 1·0366, is formed by the incomplete combustion of nitro-glycerine explosives. It is unstable in contact with air, and by taking up more oxygen becomes converted into nitric peroxide. To avoid incomplete combustion the explosive should be in good condition and the detonator of sufficient strength. Nitric oxide is slightly soluble in water.

Sulphur dioxide, sp. gr. 2·2369, can be recognized by the characteristic odour of burning sulphur. It is given off by sulphides, especially during underground fires. There is little danger of being poisoned by it, as it is so irritating that the victim, if not trapped, is forced to seek fresh air.

Sulphuretted hydrogen is known by its odour, resembling rotten eggs: it is produced from iron sulphides in the presence of water. It burns with a blue flame and explodes violently when mixed with seven times its volume of air. It is extremely poisonous, even in small quantities. It is sometimes confined in old workings by the pressure of water, and if tapped by a borehole the issuing gas might cause a small explosion in the presence of a naked light.

Cyanide fumes is another source of gassing. If tailings from cyanide vats are used for filling stopes without being thoroughly washed beforehand, cyanide poisoning may result.

Quicksilver miners are liable to salivation, but only when metallic mercury is present, mercury sulphide being one of the most insoluble sulphides. Lime-sulphur or calcium polysulphide coats the mercury with a sulphide film.

Safety Lamps. Only electric or oil safety lamps should be allowed in any gassy mine. Before a safety lamp is given out, the lamp man in charge of the lamp room should see that it is properly assembled, cleaned, trimmed, and locked. Every safety lamp should also be examined underground before the men proceed to their working places.

Both officials and men in a gassy mine should be searched for matches, tobacco, and tobacco-pipe before proceeding to their working places.

No person should hang or place a hand safety lamp within two feet from the swing of any pick, hammer, or other tool.

Should anything happen to any oil flame safety lamp to cause it to be unsafe, it must instantly be put out by drawing down the wick with the pricker or other appliance.

No person to whom an oil flame safety lamp is entrusted should interfere with it in any way beyond the necessary trimming of the wick by the pricker or other appliance provided for that purpose, except to put it out.

Should any person get into firedamp while using an oil flame safety lamp, he must not attempt to blow it out, but should draw down the wick, avoid jerking the lamp, and remove it steadily into fresh air. If the gas flares in the lamp he should smother it.

Sufficient electric or oil safety lamps should be kept underground in case any in use become damaged or extinguished.

Temperature, Humidity, and Dust

Ventilation is not only concerned with gases encountered in a mine, it also has to do with excessive temperature, humidity, and dust. Heat is increased by rock temperature with depth, chemical action, and animal heat. The increase of rock temperature varies in different localities. Under ordinary circumstances, when out of the influence of surface temperature, it increases 1° F. for every 60 to 70 ft. When under a large body of water it may be 1° for every 100 ft, while near volcanic rocks the temperature may rise 1° every 33 ft. A man working in a hot mine must sweat well. Sweat contains about 0·25 per cent sodium chloride. This loss of salt must be made up, which can be done by adding a small amount of salt to the drinking water. It is not so much the temperature that affects man as the humidity. The point of saturation varies with the temperature. Moisture in the air is measured by wet and dry thermometers. Readings are taken till they remain constant; the difference between the two thermometers is the basis for the calculations of the degree of saturation of the air. To save calculations Glaisher's tables are used. A variety of dry and wet thermometers is an instrument known as the Hygrodeik (Fig. 54) which has a very convenient graph arranged on its stand from which one can read the humidity at a glance. If both scales read the same, then the air is saturated. If the air is saturated, the evaporative power is nil, and a man who is sweating is not cooled; the perspiration just drips off him. If the air is still and hot, the human body is enveloped in a film of air at or near blood heat, and in such stagnation there is no evaporation of sweat to produce a cooling effect as it is saturated, but if the air is in motion (unless saturated) evaporation occurs, and the cooling effect takes place.

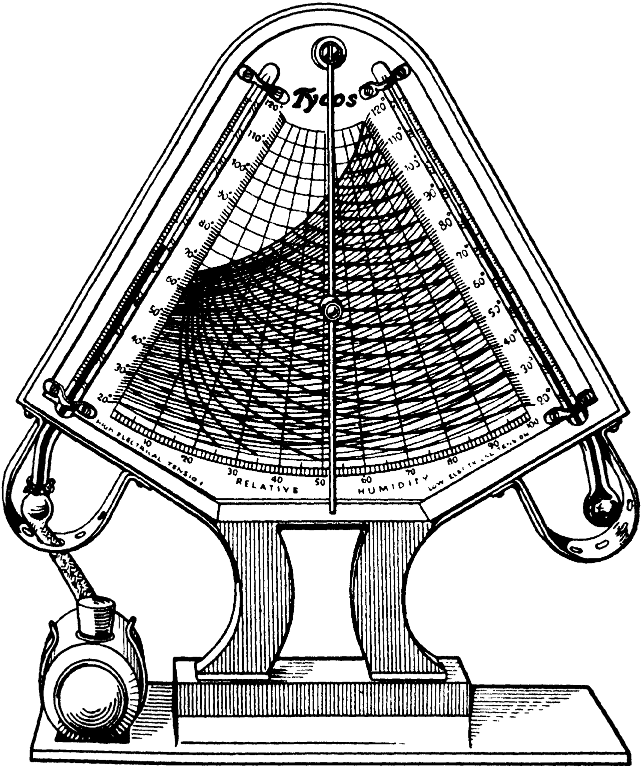


FIG. 54. HYGRODEIK

The kata-thermometer is an instrument for determining the cooling effect of air in millicalories. (A calorie is the heat required to raise 1 kilogramme of water 1° C.) The instrument is a spirit thermometer, at the top of which there is an enlargement. There are two marks etched near the top of the stem about $3\frac{1}{4}$ in. apart representing 100 and 95° F., the mean of which can be taken as the skin temperature. The instrument is taken to the place where the air is to be tested, and the bulb inserted in a thermos flask at a temperature of 150° F. until the coloured spirit fills the enlargement at the top of the tube. It is then taken out, the bulb dried, and when the spirit comes down to the upper mark, the time is noted with a stop watch how long it takes to reach the lower mark in seconds. Each kater has an individual factor of three figures marked on it, which is approximately 500. This factor divided by the time taken by the spirit to fall the specified distance in seconds gives the kata reading.

EXAMPLE. Factor 486: time 4 min 5 sec, or 245 seconds.

$$\text{Kata reading} = \frac{486}{245} = 1\cdot 9$$

Both dry and wet readings are taken. In the case of the wet, a muslin or silk net is slipped over the bulb, and after heating in the thermos, the surplus water is shaken off. The dry kata shows the cooling effect due to radiation and convection: the wet kata gives the effect due to radiation, convection and evaporation, the difference between the two being the evaporative effect of the air. Men when stripped can work with a dry kata of 6, and a wet kata of 18.

Certain dusts, such as quartz and sericite, injure and weaken the lungs of those who breathe them in, making people liable to lung troubles. Coal dust, though breathed in greater quantities, being organic, is not so dangerous to health and is eliminated after a suitable time when away from the dust. Coal dust, however, is an explosion hazard. Concerning quartz dust, Nature makes provision for preventing the larger particles from entering the lungs. It is those particles of 5 microns or less that are harmful. (A micron is the $\frac{1}{1000}$ part of a millimetre.) A common way of determining the amount of dust in mine air is the sugar-tube method, but this gives all the dust in the sample including that over 5 microns in diameter, for when a screen is used, it cannot separate particles less than 40 microns. The konometer is an instrument for causing a known quantity of air to impinge at a high velocity through a fine nozzle on the surface of a glass slide which is thinly coated with vaseline. A piston actuated by a spiral spring is contained in a cylinder of such a size that on release of the spring, which is held compressed by a trigger catch, it moves through a space equivalent to 5 c.c. The dust is deposited in a circular spot usually 0·6 to 1·0 mm in diameter. The piston speed of konometers varies between 30 and 175 m/sec. A velocity up to about 30 m/sec will not produce adherence of all the fine dust, while a velocity of 175 m/sec causes the vaseline to spread, unless the layer is very thin. Velocities between 50 and 120 m/sec give satisfactory results. The slide is examined under a microscope and the dust particles less than 5 microns counted. The diaphragm of the eye piece is ruled with two lines radiating from a centre line at 18°, thus obtaining two sectors, each representing one-twentieth of the field. Other parallel lines are ruled on the glass to help the observer to judge the size of the particles.

Since coal dust plays an important part in mine explosions, it is important to determine the quantity of dust floating in the air. To do this a glass tube about 12 in. long and 2 in. diameter has one end drawn out to a smaller diameter, while the other is slightly widened out. The tube is filled for about two-thirds of its length with cotton wool. A vessel that will hold a known quantity of water, say 3 gallons, has a hole at the bottom closed with a rubber stopper, and another hole at the top, the rubber stopper of which has a hole bored through it, into which the thin end of the glass tube passes. When the lower stopper is removed, the water passing out acts as an aspirator causing a specified quantity of air to pass through the filter. The difference in the weight of the tube and its contents before and after the operation determines the amount of dust contained in the quantity of air aspirated.

If a sample of mine air is wanted for analysing its gas contents, take a Winchester quart bottle and see that it is clean and dry inside. When at the place where the sample is to be taken, pump mine air into it through a large syringe with a rubber tube leading to the bottom of the bottle. A few strokes will soon drive all the surface air out (as can be proved by testing with a drop of HCl and NH_3). The bottle is then closed with a glass stopper and labelled. This is better than filling the bottle with water and allowing it to pour out underground so giving place to mine-air, as the moisture will absorb a certain amount of carbon dioxide.

It is generally considered that a man underground requires 20 cu ft air per minute in a metal mine, but it is usual to provide him with 50 to 100 cu ft per minute. In highly timbered mines it may have to be increased to 200 cu ft. In a coal mine where there is more oxidizable matter exposed than in an ordinary metal mine, the amount of air allowed for each person underground shall be not less than 150 cu ft; and for a horse 300 cu ft per minute.

If the ventilation of a coal mine has been suspended, the fan should be kept in operation for at least 24 hours before men are allowed to enter the mine.

Where a colliery is divided into districts, not more than 70 men should be allowed to work in each district.

The intake air should be free from the atmosphere of stables or old workings.

The quantity of air should be measured at least once a month in the main airway intake, as near as possible to the downcast shaft or tunnel mouth; also in every split as near as practicable to a point within one hundred yards out-by of the first working place which the air enters. The neighbourhood of the place where the measurement is made should be straight, uniform in area, and free from obstructions.

Wet and dry readings of the thermometer should be taken of the atmosphere at least once a month in the working place of each ventilating district; first at the intake end of the ventilating district, and secondly on the return end of the ventilating district. The highest readings should be entered in a book for that purpose.

At intervals not less than one month in the case of a gassy place, and in other cases not less than six months, the percentage of inflammable gas in the air of each district return airway should be determined. A place should not be deemed fit for working in if the air contains less than 19 per cent oxygen or more than ¼ per cent carbon dioxide, and the percentage of inflammable gas in an airway is more than ¼ per cent.

Permanent stoppings and air-crossings should be substantial in structure, reasonably air-tight, and as far as possible explosion-proof.

Where doorways are required on a road normally used for traffic between intake and return airways, double doors of substantial structure should be provided.

No fires should be used underground in any coal mine.

Dust Prevention. No hole should be drilled by machinery underground unless water is fed into the bottom of the hole, or a jet or spray of water is directed into or about the hole in such a manner and to such extent as is necessary to prevent the issue of dust from the hole during drilling operations.

In coal-mines water sprays should be used on the cutter bar of all coal-cutting machines.

Steps should be taken to allay dust produced during blasting and other dust-producing operations.

Before firing a shot in a coal mine, the face should be thoroughly wetted.

To prevent dust from entering the downcast shaft of a coal mine, no plant for screening or sorting coal should be nearer than 80 yd of the shaft.

Skips should be made of metal, except in such cases where the water is acid and would affect the metal.

Skips should be sprayed with water before coal is placed in them, and again after they are loaded.

Rock dust used for dusting in coal-mines is mostly ground limestone or shale. It should not contain more than 3 per cent total silica, and should be of such a fineness that when dry it will pass through a 60 mesh sieve per lineal inch, and not less than 50 per cent by weight should pass through a 240 mesh sieve. The mixture of incombustible dust and coal dust should be spread the full width of the roadway for several yards in length. Dust counts and analysis of samples of dust in suspension in the air should be made at least once every six months.

Circulation of Air

Heated air tends to expand and rise, while cooler air, being denser, sinks. This is the principle on which natural ventilation acts. Diffusion is the tendency of two or more gases of different densities, but originally of like temperature, to become uniformly mixed. Convection is the tendency of two currents of different temperatures to become uniformly mixed. Diffusion and convection together explain why foul gases become diluted in the absence of ventilating currents, so that in time one can breathe where a shot has been fired.

Air sometimes circulates round certain parts of a mine without leaving it, and necessarily becomes very impure, such as when hot air from deeper levels rises through passes and worked-out ground into cooler parts of a mine from which it finds its way into the downcast shaft, without reaching the surface. Old workings no longer required to be ventilated should be sealed off. Ample air may pass through drifts and cross-cuts to ventilate a mine, and yet the air in working places, where it is most wanted, may be almost stagnant. Air currents follow the line of least resistance, consequently dead-ends are passed by unless air is specially directed into them by doors, curtains or brattices.

An anemometer is used for measuring the speed of an air current under ordinary conditions. It will record velocities ranging from 200 to 3000 ft/min, but it is not suitable for low currents. Being subject to dust, moisture, and often rough usage, the instrument should be calibrated at least once a year. Every instrument has its own characteristics. Nominally a correction for velocities between 300 and 900 ft/min is under 1 per cent. Choose if possible a time for taking measurements when mine activities have ceased, so that there shall be no interference by traffic in the neighbourhood. Select a place where the cross-section is regular and the passage straight and free from obstruction. The observer flattens himself against a wall if there is no recess in which he can stand. As there is more friction of the air next the walls, roof and floor than in the centre of a passage, the best way is to attach the instrument to the end of a stick (which does not take up much room) and move it slowly in a zigzag way from wall to wall and roof to floor, taking not less than two minutes over it so as not to agitate the air. Care must be taken that the instrument is properly presented to the current, and that the movement is steady and regular. When the instrument is in good condition and properly manipulated and timed, and used at a suitable site, the error for normal velocities should be less than 5 per cent. Before starting the test, the anemometer should be kept going at a speed corresponding to the velocity of the air, and when ready to record, the dial is set in motion. The instrument gives a direct reading. If there is no cover for the operator, he may measure the air current of one half vertically first, and then change over and measure the other half. To determine low currents, the time taken for smoke from a burning fuse or odour of some strong scent to travel a given distance, or to walk with a lighted candle against the current at a rate sufficient to keep the flame straight, gives a rough indication: but a kata-thermometer gives a more accurate result. Different formulae have been devised for determining velocities with the kata-thermometer. The makers of the kata usually adopt the following -

$$V = \frac{197\left(\frac{H}{8T} - 0{\cdot}15\right)^{\frac{5}{4}}}{1{\cdot}85 - \frac{H}{T}}$$

Where V = air velocity in feet per minute
H = the cooling power as obtained by the kata
T = the difference between the mean kata temperature and that of the air, or $97{\cdot}5 - t°$ F.

For large deep mines air-conditioning plants are sometimes erected, the object being to make the air more suitable for comfort and health by cooling it, decreasing the humidity, removing dust and supplying fresh air. Air-conditioning plants are large and expensive, but are partly repaid by getting more efficient work from the underground workers.

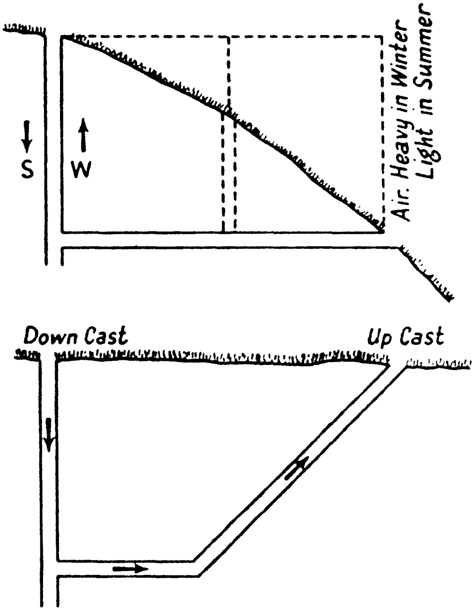


Fig. 55. Natural Ventilation
S = Summer; W = Winter

Natural Ventilation

Where artificial means are not used, the temperature in shallow mines is cooler in summer than at the surface; while in winter it is the opposite. In deep mines, the underground temperature is greater than at the surface all the year round on account of increasing rock temperature with depth. This assists ascending currents, and increases the ventilating efficiency of an upcast. Natural ventilation is often insufficient, and not as easily controlled as artificial ventilation. In places where there is a great difference in surface temperature according to the season, or even between day and night, and the difference between the levels of entrances to a mine is not great, the direction of the ventilating current may be reversed. Natural ventilation does not act well in hot climates where the outside temperature is about the same as that underground. While the air current is being reversed, there is a period of stagnation. When work is going on in a drift, the hot air rises and passes along the roof until the grade of the drift, if long, prevents it from reaching the mouth of the drift. If an adit and shaft are connected, when the temperature of the air on the surface is above that of the mine, the current will pass down the shaft and out of the mouth of the adit or a lower shaft. When the temperature at the surface is below that of the mine, the cold air passes through the adit or lower shaft where it becomes heated and escapes out of the higher shaft. This is due to the difference in weight of the two columns of air. When a vertical and an inclined shaft from the same horizon at the surface reach the same level below, the vertical shaft becomes the downcast and the inclined shaft the upcast, for being longer, the air has an opportunity of becoming warmer. Two mines with one shaft each may be connected to assist the ventilation: but this is disadvantageous for one of the mines as the farther the air travels the more it becomes vitiated, and the mine which becomes the upcast not only has its own foul air but also that of the downcast mine, and consequently finds it difficult to obtain labour when men can get employment in the downcast mine. Air passages should be sufficiently large to reduce loss by friction. It is not advisable to conduct the same air right through a mine of any size in one current, as the velocity would be too great, it would tend to dry the air and carry dust in suspension. By splitting the fresh air up for every one or two levels, it can be conducted to a number of working places. The usual current of air in a metal mine is 20–40 ft/min; if too fast it may blow naked lights out, especially when walking against the current.

Ventilation may be assisted in some cases according to needs. Shallow shafts may have fresh air delivered to those working at the bottom by a windsail. If the upcast shaft does not draw well, it may be lengthened at the top by the erection of a chimney on one side, to which it is connected by a flue, and if that is not sufficient, the air may be heated by a furnace at the surface. Pipes made of galvanized iron 10 to 20 gauge and from 10 to 16 in. diam., canvas, light tongued and grooved timber are used to draw off foul air from a dead-end or to deliver fresh air. Bratticing may be done with brattice cloth (jute), or boards nailed to a wooden framework. Brattice cloth is mostly used in collieries; the length, however, is limited on account of air leakage. Curtains hang from the roof and are made in two pieces, they overlap where they meet and are loose at the bottom so that truckers can pass through without interfering with the ventilation to any extent. Wooden doors are used, especially in coal mines, to divert currents. An air lock with a door at both ends is used when it is necessary to draw several skips or tubs at a time. The doors are made self-closing and sometimes self-opening when rope haulage is used. In other cases boys known as trappers attend to the doors and see that they are kept in order. The doors have a small slide let into them which is opened first to equalize the pressure inside and outside the air lock. If it is desired to split a current and divert a certain amount of air to a district, a small sliding door is let into the main door, and this is opened sufficiently to permit the required quantity of air to pass through. At certain places in collieries, it is necessary for currents to cross each other at right angles: this is done by overcasts or undercasts, generally the former.

Artificial Ventilation

This is necessary for large mines, and is done mechanically by blowing or suction machines. Blowing machines are mostly used in metal mines, and suction machines in colleries. If suction were used in metal mines, since there are generally several connections with the surface, they would draw air from the shallow workings in preference to those deeper down. Air from compressors is never used specially for ventilating purposes as it would be too expensive, but the exhaust from rock drills helps to ventilate working places, and as it expands, also helps to cool the air. After firing, compressed air may be turned on full for a few minutes to drive away the fumes. Small electrically driven booster or auxiliary fans help to drive air along which has slowed down owing to friction. An exhaust fan system should have the fan controls in fresh air, and the fan should be on one side of the upcast shaft or entry in case of an explosion. The downcast shaft should have no inflammable material near it in case of fire, when the products of combustion might be delivered below.

The useful effect of a fan is found by ascertaining the quantity of air put into circulation, and determining the pressure with a water gauge, when—

The H.P. in the air

$$= \frac{\text{Quantity of air in cu ft/min} \times \text{water gauge in inches} \times 5{\cdot}2}{33{,}000}$$

(5·2 is the weight of a column of air in lb/sq ft at sea level.) Large fans are driven by ropes in preference to belts, for if one rope broke the fan would not stop as it would if a belt broke. The high speed at which modern fans can be driven makes it possible to use much smaller fans than those of the Walker and Waddle types.

A water gauge is used for reading the pressure of air. As one cubic foot of water at 62° F. and under 30 in. barometric pressure weighs 62·335 lb/sq ft, each inch in height $\frac{62{\cdot}335}{12}$ equals 5·196 lb/sq in., generally taken as 5·2 lb. Davis's gauge consists of two straight pieces of boiler gauge glass, sealed into a brass block in such a manner that they are readily removed if broken. The two glasses are connected by means of a small hole in the brass block, and in this is a tap which can minimize oscillation of the water column due to the pulsations of the fan, so that the height of the water column can be read. The scale behind the tube, which is adjustable, is divided into inches and decimals of an inch. The top of each tube is open, but one is bent at right angles and passes through the partition between the two currents of air. The difference between the two levels of water in the tubes shows the difference in the air pressure in the two compartments.

The Pitot tube is also used for determining the force of a fluid. Originally used for testing the velocity of water, it consisted of an L-shaped tube. The horizontal limb faced up stream, the vertical arm pointed upwards above the water. The height of the water in the vertical limb indicates the force of the stream.

When employed for testing the force of a current of air, the apparatus consists of two tubes, open at their upper ends, the lower ends being connected with the top of a water gauge. One of its openings is directed against the current, while the other has its opening with the current. The first measures the sum of the static and velocity pressures, which are known as the total pressure of the air. The other measures the static pressure of the air; the difference measured in the water gauge is the velocity pressure.

Fans may be used for pressure or exhausting. There are many different types of rotary ventilators such as the Walker, Waddle, Capell, Schiele, Sirocco, and Turbo, which vary in shape and number of their vanes. The modern practice is to use small high-speed fans in preference to large slowly moving fans: their efficiency is equal if not better than the latter: their first cost, and that of the foundations, are also less. The work done by a fan depends on the speed of the periphery, no matter whether it is a large slowly moving, or small fast running fan.

When a certain quantity of air is required in a mine, allowance must be made for the loss of power due to friction which increases with the length of the air passage. It takes a greater quantity of warm air to balance the same weight of cool air. The rubbing surface of a passage is the length of the perimeter multiplied by the length of the air passage. Resistance to a current of air is due to it rubbing against the sides, top and bottom of the passage, turns in the airway, sudden contraction and expansion of the airway, moving and stationary objects in the airway, and to regulators. The coefficient of friction in decimals of an inch of the water gauge in—

Unlined galleries is from 0·00087 to 0·00123.
Arched passages is from 0·00030 to 0·00062.
Timbered passages is from 0·00147 to 0·00241.

Explosions

Fans on the surface should be situated on one side of the entrance to a mine, so that it shall not be damaged in case of an explosion. Mine fans should be made reversible, but should not be reversed except on the order of the officer in charge of operations; for in case of an explosion extinguishing lights, miners have only the current of air to guide them to the downcast shaft.

Sparks from low-tempered steel will not cause an explosion; but sparks from high-tempered steel may. Picks are generally low-tempered.

Underground refuge chambers of suitable size for the number of men likely to use them should be excavated in solid ground in mines liable to explosions, and should have one air-tight door. It should be provided with a barrel of fresh water and concentrated food for three days, and cylinders of compressed air to last three days. If possible the chamber should be connected with the surface by telephone. All men in the district should be familiar with the locality of such chambers. In time of danger, as soon as the last man reaches the chamber, the door should be closed, and the men await rescue.

Explosions in a bituminous coal mine may originate with coal gas, or coal dust: but the two generally combine. The old way of igniting blowers should, of course, never be resorted to. The only way to get rid of fire damp in working places is by ventilation. Gas may be withdrawn from gassy coal seams in advance of the workings by tapping it through bore holes. Pipes are sent a short distance through the bore holes and then connected with blowers which exhaust the gas. Not more than 0·5 per cent of methane must be allowed in the mine air. Methane becomes inflammable when 5·55 per cent of it is present in the air, and explodes with 7·14 per cent, increasing in violence up to 9·46 per cent, after which the intensity decreases, and ceases altogether at 16·67 per cent.

There are two ways of laying the dust: by watering or by dusting.

If coal dust is watered, it must be done thoroughly to be of any use; and the watering must be frequently carried out, as the current of air soon dries it. The watering of floor, roof and sides cannot be used in some mines, as it would cause the floor to heave, the roof and sides to crumble, and the timber would fall in.

The rock dust generally used is shale or limestone, which is pulverized so that 100 per cent will pass through a sieve of 20 mesh to the linear inch, and of this 50 per cent should pass through a 200 mesh per linear inch sieve. The coal dust becomes non-explosive when mixed with 50 per cent rock dust. There are two methods of applying rock dust: the barrier, and the coating. Rock dust may be applied by hand, or by a hand-operated blower. The latter forces the rock dust more effectually into crevices and mixes it up better with the coal dust than when done by hand. When rock dust barriers are installed, at least 100 lb per square foot of average cross-section of the entry at the barrier zone should be used. When an explosion takes place, it stirs up the dust, forming a screen which prevents the flame from propagating beyond the immediate area of origin. Samples of the dusting mixture should be taken from time to time to ascertain if that part of the mine requires redusting. It is not necessary to redust so often as to re-water.

A hole should never be tamped with coal dust. Only safety explosives should be used in breaking down coal. Not more than one shot should be fired at a time when blasting coal, and the air should be tested for firedamp after blasting. Excess of dust should be removed for several feet from where a shot is to be fired. This dust may be removed by the use of powerful suction apparatus.

Coal-dust is formed at the working face; shaken from tubs when travelling on roadways; and blown from them by the air current while hoisting. To prevent this the tubs should be dust-proof and watered both before and after loading.

The first thing to do after an explosion is to see that the ventilation is restored; then that rescue parties are supplied with proper appliances for entering the mine. The number of men in a rescue party should be checked when they go in and come out. The men work in relays. Necessary material must be got ready for subsequent operations. Send for ambulance and doctors. Notify the Government Inspector of the district, the owners (if the coal mines are not State-owned), and neighbouring colliery managers.

DRAINAGE

The system of drainage adopted for a mine has generally grown by degrees because full data are not available at first. A mine which may be wet in the upper levels may be dry at a depth on account of the compactness of the rock and the greater temperature.

There are many points to be considered, such as the nature of the water, whether acid, coppery, or gritty; the quantity of water to be handled; whether constant or periodical, due to surface floods, or occasional tapping of water courses or old workings; the probable life of the mine, and the money available for the plant; whether the drainage can be done naturally through adits, or whether water has to be raised through a shaft; the nature of the country rock, as some hold more water than others, such as limestone and vesicular basalt; whether pumping is done from one shaft, or from several.

When the country is drained, the hydraulic gradient is steep near the bottom of the shaft and flatter higher up. A mine which is sunk in advance of its neighbours, drains the latter, and some managers delay sinking to save the cost of pumping, which leads to law suits. In the early days of alluvial mining when claims were small, it was possible to keep down water as there were so many pumping or bailing, but now that leases are larger, one pump has to drain a large area, and it is not so easy to keep the water down. The cost of having to pump so many tons of water for one ton of ore mined has forced some mines to close down, which under other circumstances would have paid well. A series of mines in the same area may be drained from the same adit or series of adits; or a combination shaft may be used to drain several adjoining mines. As to whether it is cheaper to go to the expense of driving an adit or water level and have a natural drainage in hilly country, or to sink a shaft and have constant expense of pumping depends on the configuration of the country and probable life of the mine. With adits the first cost is the main item, the upkeep being nominal. Drains in adits may be cut at the sides or in the middle. In the latter case the drain is boarded over. Dirt is liable to work its way in between the boards, so has to be cleared out occasionally; and should a board break, a man may break his leg.

Controlling the Flow of Water

When sinking a shaft, it is considered wet if it makes more than 1000 gal h. Men working in wet places should work shorter hours and continuously, instead of stopping for crib in their wet clothes and risking pneumonia. When possible, water should be prevented from entering a mine. If water drips down a shaft, it may be trapped in water rings and led down pipes to a tank or sump from which it can be pumped. A water ring has the shaft shorn back at the top of it so as to lead the water in, and the ring is given a slight inclination to the exit pipe. The actual ring may be of wood, concrete, or iron. Water which enters a shaft through a crack, may be stopped by caulking it except in places where pipes are inserted through which cement is pumped under pressure. For fear the major portion of the cement might be washed away before it has time to set, a mixture of 3 sacks of Portland cement, 6 quarts of bran, and 6 quarts of oats has been used with success. All water should be caught as soon as possible and dropped through pipes (diamond drill holes have been used where the water is coppery) to the nearest pump level.

If water is expected to be encountered when boring near old workings underground with a diamond drill, the first three or four feet are drilled large enough for a pipe in which the drill can work freely. This is firmly wedged or cemented in. The inner end of the pipe is enlarged by having a coupling screwed on to keep it from being forced out of the hole by the pressure of water when it is struck. At the other end, which

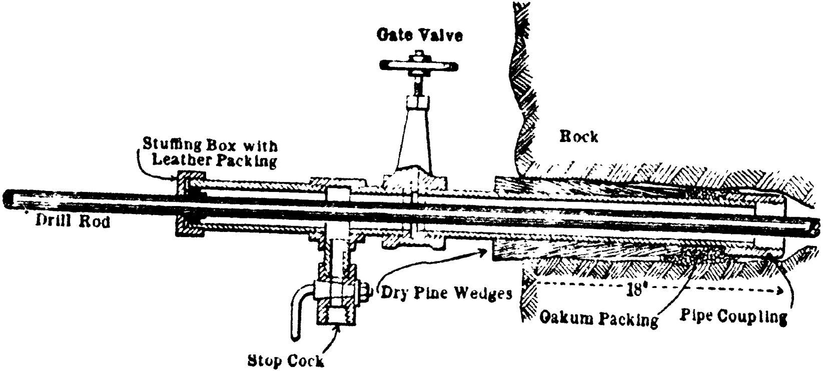


FIG. 56. ARRANGEMENT FOR CONTROLLING FLOW OF WATER TAPPED BY A DRILL HOLE

projects outside the hole, is a leather stuffing box. Between this and the rock is a gate valve; and between the outer end of the pipe and the gate valve is a short branch pipe with a stop cock which is left open to allow water and chips from the drill to escape. When water is struck, the drill rod is withdrawn, and when past the gate valve the valve is closed to cut off the water, or left open sufficiently to allow as much out as the pump can handle.

Dams

In deep alluvial mines, the main shaft is sunk outside the alluvial and well below the bottom of it. A cross-cut is driven approximately to the centre of the deposit and then a rock drift in the direction of the course of the old river bed. From this are driven branches right and left, and in the roof of these branches are put up bore holes to determine the position of the alluvial and to drain it. The mouth of the bore hole is cased and provided with a quick closing valve so that water can be cut off as soon as tapped, or its outflow regulated. In case of an overwhelming rush of water, lock-gates (Fig. 57) are provided in the drift. Such a door is about 6 ft high and 4 ft wide, consisting of steel plates ⅜ to ⅞ in. thick, strengthened by H-girders or angle-iron. The door frame is secured in strong concrete or brickwork abutments. Heavy wrought-iron hinges connect the door with the frame, the holes in the hinges being sufficiently large to allow the door when closed to be drawn up close, and a tight joint is made by rubber let into a recess. The walls are wide on the inside of the drift, but narrower on the outside. Before closing the door, the rails are taken up. Near the bottom on each side of the gate a 10 in. outlet pipe is built through the abutment provided with a wedge valve. This serves as a drain under ordinary conditions. An air pipe is also built into the abutment near the top to allow the escape of air displaced by the water which can also be closed by a valve. A ring-bolt is fastened to the centre of the outside of the gate. One end of a rope can be attached to this, and the other end is drawn tight and attached to a cross piece of timber resting against the outside of the abutment.

If an inrush of slurry is likely to be encountered, a sand-lock is installed. This consists of two wooden frames 18 by 8 in. placed 3 ft apart attached to brick abutments with 1¼ in. bolts which extend through both frames. The door is made in two parts of 18 by 6 in. hard wood, each being hung

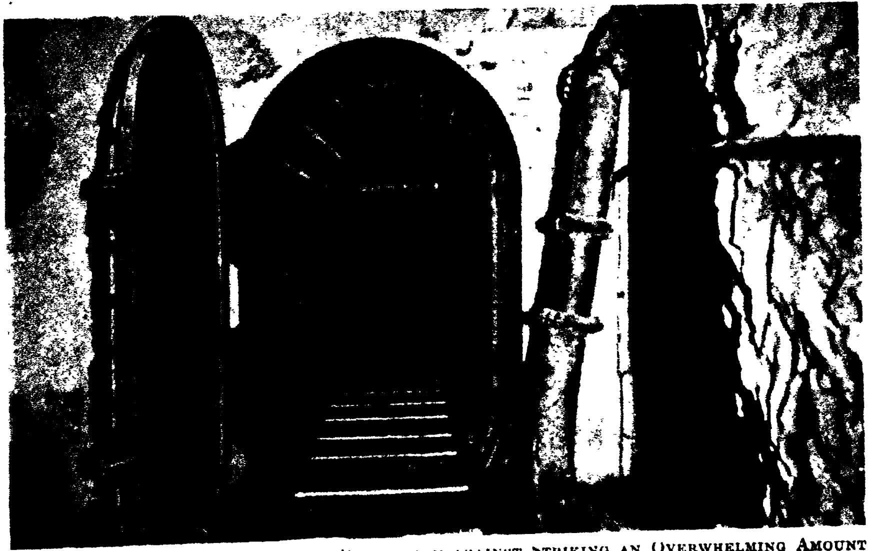


Fig. 57. Dam Door Erected as a Precaution against Striking an Overwhelming Amount of Water during Development Work

on two vertical hinges. The two parts are bevelled where they meet. There is no fastening to these doors as in the case of water-locks, the pressure of the slurry being sufficient. There is a drainage pipe passing through the abutment near the floor on each side of the gate with a 16 square ¾ in. plate held back with a wire in such a way that in case of a rush of slurry the wire will give way. The first man to escape closes the bottom section of the gate to dam back the slurry, leaving sufficient space for other men to escape over the top. The last man closes the top section which then seals that part of the mine. A 6 in. air pipe is built in the abutment near the top which has a valve on the outside that can be closed when slurry reaches up to it.

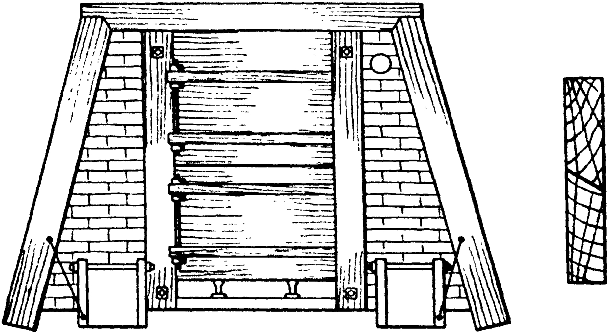


FIG. 58. SLUDGE GATE

Underground dams may be temporary or permanent. The temporary are mostly makeshifts for use while sinking another section of a shaft, when the water which would otherwise pass into the sump is diverted behind the dam which is generally built in a convenient cross-cut or drift from which the water may be pumped or led to a tank. The rock at the site for a dam should be free from cracks, and the seating should be hand-trimmed, not blasted. Temporary dams are generally built of timber or clay, or both. A simple wooden dam may be built by cutting recesses in the sides of a drift and placing horizontal baulks of timber across, the ends of which are wedged against the sides. The joints between the logs are caulked; or, there may be a double wall of timber, with a core of puddled clay between about 10 in. thick. A simple and cheap kind of clay dam is to fill old cement bags with clay at the surface with their mouths tied up in which way the material is easily conveyed underground. All loose material having been removed from the spot where the dam is to be constructed, clay is rubbed well into the surface of the rock, and the first row of bags laid closely packed on this foundation with their mouths towards the water side of the dam. The junctions between the bags are covered with clay and the usual drain pipe placed in position. Another layer of bags is placed on the top of the first, and this process is continued till the wall is high enough. The mouths of the bags are now plastered back against the face of the dam. If one wall of bags is not sufficient to withstand the pressure, another can be placed behind it. The wall may be curved so as to present a convex surface to the pressure. The pipe passing through the dam may be plugged or provided with a valve.

Solid wooden dams can be used for pressures up to 300 or 600 ft of water. They are not suitable for very great pressures as water is apt to be forced through the rock. If the rock against which the dam is to be placed is rough, it must be made smooth with cement. The timber is prepared at the surface, fitted together, and marked. They vary between 4 and 8 ft long, depending on the pressure expected. The pieces are squared and tapered in section, the curve towards the water having a radius of 18 to 30 ft. The logs are laid in layers; the horizontal joints are in line, but the vertical joints are broken. The taper of the logs and the angle of the horizontal joints are made to conform to the sphere of which it forms a part. All joints between the logs and between the logs and the rock are made of tarred brattice cloth. The last log placed acts as a key and is driven in with a sledge hammer or battering-ram. A large hole is bored in one of the pieces in the second row from the bottom to drain off any water that may collect during the construction. In the middle of the dam is a flanged cast-iron pipe large enough for a man to crawl through, say 18 in. A 1 in. hole is bored through one of the pieces in the second row from the roof and a tube placed in it with the inside end turned up to allow air to escape as water accumulates behind the dam. When water comes out of this pipe, it is plugged from the outside. The joints between the logs and round the pipe are made watertight by driving wedges into them obliquely. The first wedges are of soft wood 12 in. long, 3 in. broad, and 1 in. thick at the head end. The next wedges are smaller and of hard wood; finally iron wedges are driven in, first on the inside and then on the outside of the dam. The inside is then coated with cement made of axle grease, tar, and slacked lime. The man-hole is closed with a wooden plug from the inside after the men have passed through. The plug has a vulcanized india-rubber ring round the larger end to ensure a watertight joint. The plug is supported on a trestle at the proper height just in front of the hole. A rope is attached to an eye-bolt passing through the plug, and when all is ready, the plug is pulled into place, and the pressure of the water, when it rises, forces the plug in still tighter. Wooden dams have an advantage over those of brick or concrete, inasmuch as they give a little with slight movements of the ground, and swell when wet; besides they are more readily repaired.

Brick and concrete dams are made of two or more concentric arches placed a few feet apart, the space between being filled with clay. The clay next the outer arch serves to fill any cracks formed in the bricks or concrete by earth movements, while the inner arch preserves the clay from being washed away. Bricks are set in cement and are laid on end as when building an arch. The drift on either side of the dam is lined with bricks for four feet. Drain and air pipes, and a man-hole are used as in the case of a wooden dam. A pressure gauge can be attached to a pipe in the plug with a valve that can be turned on and off when desired to test the pressure behind the dam. It is sometimes necessary, more especially in a colliery, to build a brick dam in a shaft in order to seal off a fire or water at a lower level. In such a case the space between the two arches is filled with concrete.

Siphons

Occasionally it is necessary to drain water from an upper to a lower level over a low barrier. This may be done by a siphon. Theoretically the greatest height a siphon can raise water is 34 ft, that being the height of a column of water that the atmosphere can support at sea level: but practically 28 to 30 ft is the greatest height it will work, and then only for a short time. For continuous work the lift should not be more than 21 ft, and the lower the better, to allow for friction in the pipe. The siphon has two limbs, one shorter than the other. The shorter dips into the water to be raised, the longer is the discharge. There is no limit to the length of the longer limb within reason. The bend in the pipe should not be sharp, for air collecting in the highest part would soon cut off the flow of water. In practice a siphon has three valves, one at each end and one at the highest point. A siphon should only have one high point. Air may enter the siphon through leaky joints, or being absorbed by water is set free by agitation. All joints should be made with red lead. To start a siphon, close the valves at either end, then prime by pouring water into it from the highest point: then close the air valve and open the other two.

Theoretically, it makes no difference whether the discharge end of a siphon is under water or not, but in practice it does, for if the siphon is not flowing full, air will enter. As a siphon works better with cold water than with warm, it works better in winter than in summer if on the surface. An automatic arrangement can be set up on the highest point to allow accumulated air to escape and water to take its place.

The amount of water a siphon will discharge is calculated from the formula for the discharge of a pipe, the head being the vertical distance between the level of water at suction and discharge ends; or if the discharge end is not submerged, then to the end of the discharge pipe. The length is the total length of the pipe measuring the ups and downs.

Let v = the mean velocity of discharge in feet per second
h = head in feet
l = length of pipe in feet
d = diameter of pipe in inches
f = coefficient of friction.

Then—

$$v = 2{\cdot}315\sqrt{\left(\frac{hd}{fl\ 0{\cdot}125d}\right)}$$

Siphons require little attention; there are no power costs, and they are comparatively easy to install. They will handle any quantity of water up to the limit of the capacity of the pipes used.

Dewatering

Bailing. Bailing in buckets or tanks is resorted to in a mine which makes only a little water. If bailing has to be carried out more than 8 hours a day, it is generally better to install a pumping plant. If a mine is only subject to periodical flooding and the pump is only used occasionally, the interest on the cost of the pumping plant might be too great. Bailing is a great auxiliary to pumping in cases of emergency, but the mechanical efficiency is less. Bailing is often advisable if the water is acid or coppery as these would quickly eat into the pump and pipes. Whether bailing or pumping is used depends largely on the influx of water, the comparative cost of working, and interest on the cost of the pumping plant. Bailing is generally better than pumping during the opening up of a mine, and buckets are more convenient than tanks while sinking, on account of the inequality of the bottom; they do not take up so much room as tanks, and do not have to be protected from blasting like sinking pumps. Tanks are used when they can be lowered into a sump of water. The bottom is made V-shaped to enable it to enter the water more easily than if flat bottomed. If the flow of water in a shaft is not over 10,000 gal/h, it can generally be handled with tanks if the shaft is not too deep. For larger quantities and for greater depths water is better raised by pumps.

Pumps. All pumping in a mine is generally done through one shaft, sometimes sunk for the purpose. It is better to have the pump too large than too small in case the flow of water increases. Sumps are lodgments for water from which it can be raised to a higher level. Endeavour to arrange stations at main levels. If this cannot be done plan them according to the power of the pump.

There are three main types of mine pumps according to their motive power, viz. steam or compressed air, the Cornish plunger pump, and electric. Occasionally we see hydraulic pumps. Chambers have to be cut out for steam, electric, and hydraulic pumps, but Cornish pumps being vertical are in the shaft, where they take up a good deal of room.

When selecting a pump one has to consider the first cost, efficiency and reliability, the probable wear and tear, the position in which it has to be used (whether in a high and narrow, or low and long place), the motive power to be used. If only one or two small plants, it may be advisable to use steam or compressed air; if several pumps are required it might be better to use electricity: but it largely depends on the kind of power that is available. Whether all the water is to be collected in the well of a shaft, or pumped from each level, depends on whether one aims at simplicity or economy. Sinking pumps should be able to run under water.

The usual practice is to allow the speed of water in the suction pipe (which is larger in diameter than the delivery pipe) to be 300 ft/min, and that of the delivery pipe 400 ft/min. The larger the pipe the less the velocity, and consequently the less frictional resistance; but then the first cost is greater.

Cisterns are generally put in 300 ft apart vertically; if at a greater distance, the wear and tear on the clacks of the pump are too great.

Pumps may be single or double acting. When double acting they have a suction and delivery pipe in common. The lower or suction pipe, through which the water is drawn, should not be more than 22 ft in length. The total area of the holes in the wind-bore must be greater than the area of the suction pipe. The wind-bore should be kept submerged. If it is not covered completely, and air is sucked in, it is said to be "in fork." If the upper portion of the wind-bore is permanently out of the water the exposed holes should be plugged with wood. When sinking, the length of suction has to be frequently altered. This is overcome by the use of a telescopic suction pipe; or if water collects in different parts of the bottom, a rubber hose with a strainer at the end may be used.

Steam Pumps. These are usually double acting. Steam pumps may be simple or compound. Simple or compound refer to the steam cylinders: single, duplex, or triplex refer to the pump cylinders. Two steam pumps placed side by side with suction and delivery pipes in common are called duplex pumps: they are usually so set that when the steam piston of one is completing its stroke, the other is in the middle of its stroke. A double acting pump made to run in this manner and provided with an air chamber of sufficient size will deliver water with nearly a uniform velocity. A triplex pump has its cranks set at an angle of 120°. Air chambers on a pump serve the same purpose as a fly-wheel does to a steam engine by giving a nearly uniform velocity; they also help to prevent shocks owing to the incompressibility of water. Air chambers should be placed as near the pump cylinder as possible. They are absolutely necessary on large pumps. Though steam pumps are much used, the great objection to them is the loss of steam due to transmission for long distances, the disposal of gaseous products, and the heat given off in a confined space is also a serious obstacle underground. Mine pumps working continuously are generally designed to have a plunger speed of 100 ft/min as this does not bring excessive strains on the pump. As all steam pumps of the ordinary pattern take steam at full stroke, the economy of steam is low. For mine pumps plungers are almost invariably used instead of pistons. Valves for mine pumps should be simple, strong, easily examined, and quickly replaced.

Cornish Pumps. The consumption of steam in high-class Cornish engines is from 26 to 30 lb per indicated horse-power. Except for very large pumps, Cornish pumps are generally driven by horizontal engines, and these are geared, as the pump rod must move slowly, while for the sake of steam economy the engine should go at a higher speed. The fly-wheel gives more even distribution of power than when a vertical engine is used. The weight of the pump rods should not be less than 20 per cent heavier than the water load. Balance bobs are put in every 600 to 700 ft to take the weight of the rods off the pin of the upper bob. The efficiency of a Cornish pump is about 79 per cent. Cornish pumps take up much room in a shaft, and are expensive in first cost, and to install, but the upkeep is small in comparison with other types of pumps and in a few years this makes up for the greater first cost.

Diam. of Plunger (in.)	Length of Stroke (ft)	No. of Strokes	I.H.P. required per 100 ft of Lift	Gal per Hour
6	6	12	6	4,000
12	8	10	24	17,000
20¾	8	8	50	40,000

The length of stroke can be altered by shifting the connecting rod to another of the three eccentric points on the engine cogwheel. If the distance between the engine and the surface balance bob is great, the connection is made by one or more sweep rods attached to swinging supports. Bobs may be in the form of an inverted T, an L, or V depending on circumstances. A good rule is to make the limbs one and a half times the length of the stroke. Plungers should be increased in size as they near the bottom to make up for the decreased length of stroke resulting from the elasticity of the pump rod. Cisterns are placed on the top of each lift and should have a capacity of at least three times that of the pump drawing water from it. One great advantage a Cornish pump has is that it can still work when drowned out.

Cast-iron pipes are not so readily attacked by acid water as wrought-iron. A cast-iron pipe may become cracked and have to be replaced. This has been done on occasions, though the weight of the rising main was several tons. Suitable supports were placed across the shaft on either side of the column every 30 ft to support the weight, and clamps were fitted loosely round the pipes below flanges. The pipes were then emptied and disconnected top and bottom. Steam was turned on at the bottom of the column through a ½ in. pipe provided with a valve for an hour or more till the pipes below the faulty section expanded sufficiently. The columns were then clamped tightly or wedged, and the steam cut off. The pipes were then allowed to cool and contract, the faulty section removed and a new one put in its place. Steam was again turned on; the pieces bolted together, clamps or wedges removed, the pipes allowed to cool, and the rising main connected up again to the pump.

Electric Pumps. These may be centrifugal or horizontal. There is less loss of power through leakage and friction than with steam pumps. An accident to the power-carrying wire can be more readily repaired than a steam pipe. There is no loss through condensation of steam, and no trouble about the disposal of exhaust steam; less space is required for wires than for pipes, and no heat is given off from them. Ordinary centrifugal pumps are limited to heads of about 60 ft, but the turbine pump is built in single units for pumping over 1000 ft. For high lifts one has to use multiple chamber turbine pumps. The selection of a pump does not depend entirely on the method of drive, the rotative speed, and number of stages, but also on the number of vanes, the vane angle, ratio of width of impeller to its diameter, and the peripheral speed of the impeller. The water after passing through the suction pipe enters the centre of the first impeller and, leaving the periphery at a high velocity, passes between stationary scroll-shaped blades called the diffuser, the function of which is to change velocity into pressure with a minimum of loss due to friction and eddies. After leaving the diffuser, the water travels inward through an S-shaped passage communicating with the eye of the next propeller, and so on till the water ultimately finds its way into the discharge pipe. The openings are a little larger in each successive step so as to reduce the friction due to speed. Each chamber contributes its share of pressure, the total pressure being represented by the sum of the pressures generated by the individual stages. Multi-stage pumps occupy only a small space and do not require much daily attention. Heavy foundations are unnecessary, the wear and tear are slight as the only moving part in contact is the steel spindle, which is well lubricated. No air vessel is required on account of the speed of rotation. These pumps require a large quantity of water to make them efficient. The efficiency of a turbine pump is not so great as that of Cornish pumps which is 60 to 65, but the first cost is less. The number of revolutions of a turbine pump are the same for a head of 100 ft as for one of 1000 ft. The speed does not increase with the load, but more power is required. Electric sinking pumps together with their motors are now made watertight in cases of steel, so it does not matter if they are submerged. They can be easily raised or lowered in a shaft.

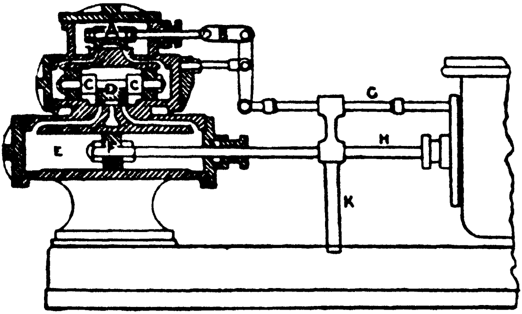


FIG. 59. EVANS' HYDRAULIC PUMP

Horizontal, electrically-driven pumps are double geared 3 : 1, as the motor has to run fast and the pump slow. In case the pump should become charged with electricity through some defect, it is advisable to connect the frame with the earth to prevent it becoming dangerous to anyone touching it.

Hydraulic Pumping Engines. These are worked on the principle that a small quantity of water falling from a great height will raise a large quantity of water a lesser height. Evans' hydraulic pump is used for raising water that has accumulated some distance from a shaft below the level of water in the sump. The hydraulic engine consists of the motor cylinder (*E*) and two valves (*A*, *D*) above it. The piston rod (*F*) of the motor cylinder is a continuation of the plunger rod (*H*). The water which drives the engine enters the chest of the jockey valve under full pressure due to head, which may be obtained from the rising main. It then passes through into the chest of the main valve; from that it passes into the motor cylinder and moves the piston forward. The exhaust water escapes through the exhaust port into the sump. Attached to the piston rod is an arm (*K*) that alternately strikes lugs on a rod (*G*) which actuates a lever, the other end of which is attached to the valve stem of the auxiliary valve by means of a link (*B*), thereby exposing alternately the water and water exhaust pipe. The motor piston is always smaller than the pump plunger, but the length of the stroke is the same. The jockey or auxiliary valve makes the dead centre almost an impossibility, and enables the pump to work so slowly that it can scarcely be seen to move.

Drainage Formulae

1 cu ft water = 62·4 lb = 6·2355 gal.

1 cu in. water = 0·036 lb.

1 imperial gal of water = 10 lb = 0·16 cu ft = 277·274 cu in.

1 ton of water = 35·9 cu ft = 224 gal.

P = Pressure in lb sq in.
H = Head of water in feet.

$$P = H \times 0{\cdot}4333$$
$$H = P \times 2{\cdot}31$$

Pressure per sq ft = $H \times 62{\cdot}4$.
Cubic feet of water × 0·028 = tons.

To determine the strength of a mine dam, let

t = thickness of the dam in inches
r = the shorter radius of the dam in inches
w = width of opening or span in inches
p = pressure of water at dam in lb sq in.
s = comparative strength of material in lb sq in.

Then for an arched dam—

$$t = \frac{pw\sqrt{4^{r-1}}}{4s}$$

The shorter radius of the dam should be from ¼ to ⅓ greater than the width of the opening. For practical reasons, it would not be advisable to build a brick dam of less than 20 in. Take 10 as a factor of safety. The crushing strength of bricks may be taken as 90 t/sq ft. but the working pressure on a brick should not be more than 14 t sq ft.

Pump Calculations.

Let G = the number of gallons discharged per minute
S = the plunger speed in feet per minute
d = diameter of cylinder in inches
H = the number of horse-power
h = the vertical height in feet between the highest point of the centre of delivery pipe and level of water in the sump.

Then—
$$d = 4{\cdot}95\sqrt{\frac{G}{S}}$$

but as there is always more or less slip of water past the plungers, it is usual to add ¼ the number of required number of gallons of water, therefore

$$d = 5{\cdot}535\sqrt{\frac{G}{S}}$$

$G = 0{\cdot}03264d^2S$ when making the same allowance for slip.

If the theoretical discharge is required, then—

$$G = 0{\cdot}0408d^2S$$
$$H = 0{\cdot}00038Gh$$
$$h = \frac{H}{0{\cdot}00038G}$$

Calculation for Steam Pumps. To find the diameter of pump plungers to pump a given quantity of water at a piston speed of 100 ft/min.

Let n = the number of gallons

Then the diameter in inches for a single double-acting pump will be $\sqrt{\frac{n}{4}}$. For a duplex double-acting pump, divide by 8 instead of by 4.

To find the number of gallons delivered per min. by a single double-acting pump at 100 ft speed per min.

Let p = the diameter of the plunger.

Then for a single double-acting pump, the number of gallons delivered per min. will be $p^2 \times 4$.

For a duplex double-acting pump multiply by 8 instead of by 4.

Balancing Pump Rods.

Let g = acceleration due to gravity (usually taken as 32·16 feet per sec)
f = acceleration in feet per sec
W = weight to be accelerated, i.e. weight of pump-rods plus weight of water in column, in tons
F = the force, i.e. weight of the rods and accessories, less weight of water column and frictional resistance in tons.

Then—
$$f = \frac{gF}{W}$$

Let t = time in seconds of one stroke
s = space travelled by one stroke in feet.

Then—
$$t = \sqrt{\frac{2s}{f}}$$

and the number of feet travelled per min =
$$\frac{s}{t} \times 60$$

by which one can see whether the rods must be weighted, counterbalanced, or are correct.

Capacity of a Pump.

Let L = practical discharge in cubic feet per hour
C = capacity in cu in. (length of stroke × area of plunger)
N = number of revolutions of pump-shaft per minute.

Then—
$$L = \frac{C \times N \times 60 \times 3}{1728 \times 5}$$

Allow $\frac{2}{5}$ theoretical capacity for slippage, etc., then as $\frac{3}{5}$ efficiency, 60 min and 1728 cu in./cu ft are constants

$$L = \frac{C \times N}{48} \text{ and } C = \frac{L \times 48}{N}$$

COMPRESSED AIR

Compressed air is much used in mining for rock drills, pumps, hoists and fans. Though air under pressure is too expensive to use for ventilating purposes, and the smell of volatilized lubricating oil is unpleasant, yet the exhaust of such machines helps to supply fresh air underground, and by expanding helps to cool the atmosphere. Compressed air is often used to clean dust out of drill holes and to drive gases out of working places after blasting, so as to save time.

Air is an elastic fluid which, when free from vapour, behaves as a perfect gas. At ordinary atmospheric pressure, and at 60° F. 13·09 cu ft weighs 1 lb.

According to Boyle's law, the volume of a gas varies inversely as the pressure affecting it, so long as the temperature remains constant: consequently by doubling or trebling the pressure, the volume becomes half or one-third respectively.

According to Charles' law, if the volume of a gas is kept constant, the pressure varies as the absolute temperature: and if the pressure is kept constant, the volume varies as the absolute temperature. The volume of a given mass of gas increases or decreases by $\frac{1}{273}$ of its volume at 0° C. for every centigrade degree rise or fall in temperature.

By the law of transmutation of energy, work performed on a body, whether solid, liquid, or gaseous, is shown by a definite increase of temperature in that body. Consequently when air is compressed, it is heated; when heated it expands, and the volume of air to be compressed is increased in proportion, requiring a corresponding expenditure of the power needed to compress it.

If the temperature of the air undergoing compression could be kept constant (isothermal), and the heat taken up from it returned to the air during its expansion in the motor while doing work, all loss from this source would be avoided: but as this is impossible, the aim of modern compressor makers is to prevent an increase of the volume of air by keeping down the temperature during the period of compression.

When no provision is made for cooling the air during the compression it is known as adiabatic. If the air is not cooled while being compressed, it will require more power to reduce it to a certain bulk than if compressed isothermally. This extra work is entirely used in heating the air which subsequently cools down to the temperature of the external air; the excess of work is therefore lost, and as the air cools, its pressure falls. The cooler the air when it enters the compressor the better.

The capacity of an air compressor should be expressed in cubic feet of *fresh air* per minute at intake temperature. The supply of air should be taken from where it is dry, cool and free from dust, and is best filtered. Dust in the course of time will account for a considerable amount of wear in the compressor cylinder. The fabric surrounding the air pipe has an end dipping into an oil well. The oil-laden fabric collects the dust as the air impinges on it. If using cheese-cloth the filter should have at least one square foot for every 25 cu ft free air capacity per min and the screen should be at least four times the area of the pipe to allow for the fibres of the cloth and frictional resistance. The intake of air should be outside the engine room, for should a fire occur in the neighbourhood of the intake, poisonous gases might be delivered to the miners below.

Stage Compression

In using a single stage compressor for 100 lb pressure, which is the usual pressure required for rock drills, the high temperature reached may vaporize the cylinder oil, unless it has a high flashing point, and with the air form an explosive mixture which ignited by a glowing piece of carbon may wreck the compressor. With stage compression, the pressure is gradually piled up. The gain in economy by compounding is estimated from 10 to 15 per cent. By stage compression, the resistance throughout the stroke is more uniform than with single compression, the loss through clearance is less, and there are two water jackets instead of one. The object of the water jacket is not so much to cool the air being compressed as to keep the lubricant, packing, piston rod, and valves cool. The piston should work as close to the cover as possible so as to make the clearance space small, as the first part of the stroke is ineffective until the compressed air in the clearance space becomes the same pressure as that of the incoming air. Only the best kind of piston rings should be used for air cylinders, because pistons of an air compressor are more difficult to keep tight than a steam piston, when moisture from the steam helps to keep the piston rings tight

If only a low pressure is required, it is doubtful if any practical economy would result from stage compression: but for pressures above 60 lb the advantages are very marked. In the first stage, the air is compressed to 35 lb, and after being cooled in an intercooler passes to the smaller cylinder where it is further compressed. Combined speed and pressure governors are used. The speed governor is necessary in case a number of drills are suddenly started at a time when the compressor is liable to race. The pressure governor is needed in case a number of drills is suddenly cut off. It is usual to estimate the amount of air required by a rock drill at 15 cu ft free air, at 60 lb pressure per min for a 3 in. drill, and 20 cu ft for a 3½ in. drill. In estimating the size of a compressor to drive six drills, assume that three drills will be working at the same time; one may be under repair, and the other two may be in process of transit or being set up. It seldom pays to install a motor-driven air compressor under 25 h.p. For 100 h.p. and over it is better to have two or three motor units. If a compressor is not capable of supplying all the drills ready to work, it is more economical to reduce the number of drills than to work more drills at a low pressure, when they all lose efficiency. Vertical air compressors are more in first cost than those which are horizontal; but the wear and tear, and loss in friction are very much less. Both air and steam cylinders may be either simple or compound, and these can be arranged in several ways.

When not more than three or four drills are to be worked, gasoline air compressors may be the cheapest in some places. Being self-regulating, very little gasoline is used when the drills are not working. It saves the wages of a fireman, also the cost of hauling firewood and water. One pint of gasoline is required per horse-power per hour.

In the case of turbo-air compressors, the impellers rotate inside a casing which has stationary vanes inside. It is just the reverse of a water or steam turbine, the fluid being acted upon by the impellers instead of actuating them. They have from 1 to 30 stages, each consisting of a separate impeller with discharge vanes and casing, each raising the air pressure a few pounds only. Its advantages are simplicity; economy of space and foundations; freedom from vibration due to steady non-pulsating discharge. It has a large overload capacity, high efficiency, low initial cost, and low cost of maintenance, attendance, and oil; as well as freedom from danger from explosion in air receiver or pipe line, because of absence of oil in the interior. They are made in sizes of 3500 to 12,000 cu ft free air per min with a discharge pressure of 100 lb. They have an efficiency of 63 to 67 per cent.

Lubricants

One cannot be too careful in the choice of the lubricant used in an air compressor cylinder. That employed for steam cylinders is not suitable. It should be a medium-bodied pure mineral oil of the highest quality, not compounded with a fixed oil such as those of animal or vegetable origin. Only the best grade cylinder oil should be used, and that in only sufficient quantity to keep the cylinder walls and valves just greasy. Actually, very little oil is needed owing to the absence of any tendency for the oil to be washed away as in the case of steam engine cylinders. A distinction should be made between those mineral oils which have a paraffin base and those having an asphalt base. Apparently all cylinder oils carbonize to some extent. This carbon is inflammable and its becoming oil-soaked is always a source of danger. Carbon deposit from a paraffin base oil is adhesive, hard, and flinty: that from oil with an asphalt base is of a light fluffy nature and easily cleaned. The flash point of a series of paraffin base compressor oils varies from 375° to 500° F. and their firing point is from 425° to 575°. The flash point of a series of asphalt base compressor oils varies from 303° to 375°, and the firing point from 360 to 404° F. There is not much of a margin between the temperature generated in the cylinders and the flash or firing point of lubricating oils.

Flake graphite has sometimes been recommended for the lubrication of compressor cylinders, as it avoids any danger of explosion. It neither produces objectionable carbon deposits nor volatilizes when exposed to heat developed in the cylinders. Graphite has excellent lubricating qualities.

Soap suds have been used with good results instead of oil for lubricating purposes: but before closing down the admission of soap suds is stopped and the compressor is run with oil for half an hour to prevent rusting.

Types of Compression

Compressed air, like steam, can be used for motive purposes; it is more convenient underground since, instead of adding to the heat and moisture it supplies additional air, and by expanding helps to cool it. Air compressors are rated according to the quantity of fresh air they consume. The air may be compressed in one, two, or three stages, according to the pressure required, with intercoolers between the stages, so as to reduce the volume and enable each stage to condense more air.

There are two main types of air compressors. The reciprocating and the rotary. The former can be used for compressing up to 8000 cu ft free air per minute: beyond this the rotary type is generally used.

The reciprocating type of compressor may have its valves set in motion by springs, or the valves may be mechanically driven as in the case of the Corliss valve, which, being positive in its action, insures full intake of free air with each stroke of the piston. Being placed in the heads, the clearance is less than with any other form of intake valve. The outlet valve may be of the poppet type, which, being light and working vertically, has little friction and consequently no tendency to wear out of line with its seat. For compressors driven by electric motors, new conditions are encountered. Most of the motors used are inductive or synchronous which run practically at a constant speed, hence the amount of air cannot be regulated by the speed of the motor. This is generally done by controlling the quantity of air entering the compressor.

The number of revolutions per minute for a reciprocating type of compressor may vary from 200 for a compressor having a capacity of 145 cu ft min to 140 for a capacity of 2394 cu ft.

Both steam and air cylinders may be either simple or compound. They may be arranged in several ways.

1. *The Simple or Straight Line Type.* The strains in such machines are direct.

(*a*) Single cylinder for both steam and air. This type is only used where high operating economy is second to simplicity, first cost, and ability to stand hard work.

(*b*) Single steam cylinder and compound air cylinders. This is simple and easily managed, but does not give the highest efficiency.

(*c*) Tandem compound steam and air cylinders. This is simple and effective.

2. *Duplex.* These are built in four combinations of cylinders. The advantage of the duplex type over the straight line type is that since the cranks are at right angles, the engine cannot get on its centre. The power and resistance being equalized through opposite cylinders, large fly-wheels are not necessary. One cylinder can be detached from the other if only half the capacity of the machine is required, as is sometimes the case at first.

(*a*) Cross-compound steam, and cross-compound air cylinders.

(*b*) Cross-compound steam, and duplex air cylinders.

(*c*) Duplex steam, and cross-compound air cylinders.

(*d*) Duplex steam, and duplex air cylinders.

The disadvantage of duplex types is that the strains are indirect, angular, and intermittent; therefore the parts and foundation have to be much stronger. The friction loss in duplex compressors is seldom less than 15 per cent of the horse-power of the engine, while with the straight line it is sometimes as low as 5 per cent.

All parts of an air compressor should be readily accessible. Compressed air always contains a certain amount of oil and water which collect in the pipes and receiver where they become offensive, unless the air receiver is regularly blown off and the pipes periodically cleaned.

Excessive carbon deposits, which are sticky in the early stages and become hard and flinty later, may be due to any of the following causes—

1. The ill-advised use of an oil, such as cylinder oil, which easily decomposes with the heat of the air cylinder.
2. The use of oils with too great a viscosity, known as "too heavy oils." These do not atomize easily and therefore remain too long in contact with the hot cylinder walls, thus baking down to a hard carbon deposit.
3. The use of too much oil which has the same effect as the use of too heavy an oil as far as carbonization is concerned.
4. The failure to provide a proper screen over the air intake of the compressor, thus allowing free access of dust.

The pressure in compressed air pipes is not sufficient to require extra strong pipes. Sharp and unnecessary bends should be avoided. Gate valves should be used in preference to globe valves as the friction is less. Pipes of less than 2 in. diameter are seldom used except for the nearest 100 to 200 ft to the drill, as the loss due to friction increases with the smaller pipe. Pipes larger than 4 in. should be flanged rather than screwed. Leaks may appear unimportant, but they are going on all the time and add to the load of the compressor.

HAULING

Self-acting Inclines*

Let a = the angle of inclination
C = the coefficient of friction of trucks and rope
W = the weight in lb of the loaded truck
w = the weight in lb of the empty truck
r = the weight of the rope for the length of the incline in lb
C'' = the coefficient of friction for the drum
f = the amount of resistance due to friction for drum in lb, or $= 2C''[(w + r)\sin a + C'(w + r)\cos a]$.

The coefficient of friction is equal to the tangent of the angle of inclination (also known as "the angle of friction," "the limiting angle of frictional stability," or the "angle of repose"), on which the force exerted by gravity is exactly counterbalanced by the frictional resistance.

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* A. Bowie, "Problems in Hauling and Hoisting," *T. Am. I.M.E.*, 1901, Vol. xxxi, p. 265.

When a wagon (W) is placed on an inclined plane, the force with which it tends to move down the plane, disregarding friction, is—

$$W \sin a$$

As the amount of friction equals the pressure multiplied by the coefficient of friction, the amount of friction encountered in moving a wagon (W) on an inclined plane is—

$$WC \cos a$$

When $W \sin a = WC \cos a$ or $\frac{\sin a}{\cos a} = \tan a = C$ the force with which the wagon tends to move downhill is exactly held in equilibrium by the amount of friction. The force with which a loaded wagon tends to move down the plane when the angle of inclination exceeds the angle of friction is—

$$W \sin a - WC \cos a.$$

and under the same conditions, the force with which the empty wagon resists motion up the hill is

$$w \sin a + WC \cos a$$

The smaller the difference between W and w the greater the angle of slope required to make a self-acting plane.

One must also consider the weight of the rope and its friction on the rollers of the incline, and the friction on the periphery and axle of the drum round which the rope passes. The principal factors in determining the coefficient of friction for wheeled carriages moving on rails are the ratio of the diameter of the wheel to that of the axle, the quality of the lubricant, and the smoothness of the contact surfaces. Take the coefficient of friction for the rope on the rollers as being the same as that of the wagon, though it should really be a little greater on account of the sag of the rope and the roughness of the surface. The resistance of the rope will be continually decreasing as the empty car ascends the plane. The required angle of inclination will increase with the length of the incline.

So long as $W \sin a > (w + r) \sin a + C (W + w + r) \cos a$, the conditions permit a self-acting plane, but when $W \sin a$ is equal to or smaller than the second number of this formula, no motion can be produced by gravity alone.

As the weights for steel ropes are nearly in proportion to their respective safe working strengths, provided the load too is increased, the weight of the rope must also be increased in the same ratio. Therefore the angle sought would be the same for any number of wagons per trip as for one wagon. But if the rope used for a one wagon trip is stronger than necessary, so that additional wagons can be put on without using a heavier rope, then it may be possible to make the plane sufficiently self-acting by simply increasing the number of wagons in a set.

As the resistance of the empty wagon and rope to the motion up the plane is –

$$(w + r) \sin a + C\,(w + r) \cos a$$

the strain executed by the loaded car to move down must be at least equal to this, hence the strain on the drum round which the connecting rope passes must be at least –

$$2[(w + r) \sin a + C\,(w + r) \cos a]$$

The tangent of the angle of minimum grade of a self-acting gravity incline, when all resistance of gravity and friction are considered, equals

$$\frac{C\,(W + w + r) \times \dfrac{f}{\cos a}}{W - (w + r)}$$

If there is not much to spare above the necessary grade, there should be a short piece of level track at the bottom of the incline and, if necessary, a heavier grade at the top, so that the wagon can start more easily. It is best practice to have the grade as nearly uniform as possible. Anyhow, there should not be too sudden a change in level, or else the wagon when passing from a steep grade to a flatter one may have the upper wheels lifted off the track by the rope. The frictional resistance encountered in starting from a state of rest may be taken as at about twice the friction of motion. If the grade is made greater at the top, and lighter at the bottom, the speed due to acceleration will diminish, as the motive force varies with the size of the angle of inclination; the inertia carries the wagons over the flat portion.

Aerial Ropeways

The following maximum figures do not necessarily hold good in every case. For instance, a heavy load will not run on a rope for the maximum span. Different systems are suitable for different conditions.

1. One fixed rope down which carriers run uncontrolled. This is suitable for hilly country. No support is required between terminals. Used in cases where the goods, such as

ore, cannot be damaged by rough handling. Good for spans up to 7000 ft. A ½ in. diameter rope can be used, fastened at the top and tightened at the bottom by a Spanish windlass or block and tackle. The ore is sent down in bags, not too tightly filled so that they will hang over their sling sufficiently not to fall off. The rope slings are suspended from a carrier. The carrier may be a 2½ in. pulley wheel, the axle of which is a piece of ½ in. iron rod one side of which is extended and bent into a hook; or a piece of wood about 2 in. in diameter with a double hook cut in it, one to slide on the rope, the other for the sling to hang from. The pulley carriers are used on low grades; the wooden carriers, which cause greater friction, on higher grades.

Should a wooden carrier hang up half way, a pulley carrier fully loaded may be sent down to push it on: if that does not succeed, it may have to be jerked off. The carriers with their loads are tripped off at the bottom by throwing a bag over the wire in front of them. Should this fail, a stop of branches is placed near the end of the wire. The track must be kept clear under the wire, so that any bags which fall off can be collected periodically. The rope or wire must of course be in one piece. It is best laid from the top so as to obtain the advantage of gravity; but if it has to be laid from the bottom end, a string of men or horses, each carrying a few coils with a few feet of rope between them, convey it up hill, laying it as they go. The carriers are collected at the bottom and conveyed to the top in bags by men or horses. The rope has a grade of 6 per cent upward. The force of an uncontrolled load may be controlled to a certain extent by the sag of the rope. Windage has to be taken into consideration especially when the grade is slight.

2. One fixed rope and an endless hauling rope. One carrier is drawn backward and forward by an endless rope. It will work on inclines 1 in 1 or even steeper, and for spans up to 6000 ft and loads up to 5 tons. The motive power must have reversing motion. The hauling rope is supported on a pulley attached to an arm outside each post.

3. Two fixed ropes and one open hauling rope with a carrier at each end, worked by gravity. For spans up to 6000 ft and loads up to 6 tons. Can run at a speed of 30 to 40 m.p.h., controlled by brake gear. When towers are used as supports the haulage rope must not travel over 10 m.p.h. It is self-acting when the grade is not less than 15 per cent. The carrier rope may be ¾ in. diameter, and the hauling rope ¼ in. for ordinary purposes.

4. Two fixed ropes with an endless hauling rope. A gradient of 8 per cent will be self-operative. Speed 3½ m.p.h. If the distance between terminals is over 15,000 to 18,000 ft there should be intermediate stations; and when the distance between stations exceed 1½ miles, there should be intermediate tightening gear. Spans may be up to 1500 ft long. Two men with scoop shovels can load about 6 tons per hour into moving buckets. With an automatic loader 30 tons can be loaded by one man. Suitable for quantities between 40 and 50 tons per hour. This system does not lend itself to sudden vertical changes. The first cost is great, but it is economical in wear and tear. Where excess of power is developed, it may be absorbed by a water brake (a paddle wheel revolving in water). If a friction brake is employed, the usual allowance of brake surface of 72 sq in. for the absorption of each horse-power generated is made. The following formulae are applicable to both multiple and two-bucket tramways.

Let W = total weight of traction cable in lb
w = total weight of empty and loaded buckets
v = speed of tram in feet per minute.

Then—

the horse-power absorbed by friction $= \dfrac{(W + w) \times 3\% \, v}{33{,}000}$

Let H = difference in elevation between upper and lower terminals in feet
s = spacing of buckets in feet
y = weight in one loaded bucket in lb
v = speed of tram in feet per minute.

Then— the gross horse-power $= \dfrac{H \times y \times v}{s \times 33{,}000}$

The net horse-power developed is equal to the gross horse-power minus the horse-power absorbed by friction.

The carrier rope for the full buckets is from 0·8 in. to 1·6 in. diameter; that for the empties need not be so large and varies between 0·6 in. and 1·2 in. in diameter. These last 5 to 10 years. The hauling rope is ⅝ in. in diameter and lasts about 8 years. The buckets hold 4 to 14 cwt and are placed at intervals of about 150 ft. The buckets are attached to the hauling rope by fixed or adjustable clips. When possible, supports for the carrying ropes should be supplied every 50 to 60 yards.

5. Endless running rope with carriers suspended therefrom, either loose or detachable. For quantities not exceeding 30 tons per hour. Individual loads not over 6 cwt, and spans not over 600 ft. It is advisable to have ropeways in a straight line. Any deviation from a straight line should be in the form of an angle. When fixing permanent clips, start at the upper terminal. Load uniformly in each bucket and distance them properly. Speed not over 175 ft min. If horizontal the carriers just rest on the rope. Friction grips are limited to grades of 1 in 5, if steeper one must use fixed carriers. When transferring buckets from one station to another, use a hanging rail with feather edges where it comes in contact with the ropes.

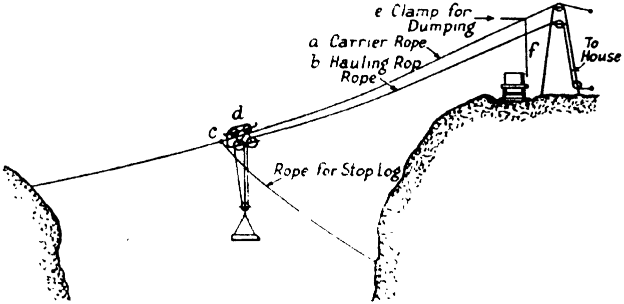


FIG. 60. FLYING FOX

Towers of wood or iron may be constructed to carry the ropes. When possible they should be spaced 150 to 300 ft apart, but if spans are across valleys or rivers, they may be up to 2000 ft. The height of the towers varies to equalize the irregularities of the ground. The arm on which the sheave is situated projects about the diameter of the sheave from the tower. The sheaves vary from 22 to 36 in. diameter. The line along the track of the rope should be kept clear for 50 ft wide for fear of grass fires. If the rope is inclined to rise, guard sheaves are used overhead to keep it down, the depression of the rope due to the weight of the bucket when passing prevents the carrier from touching it.

6. *The Flying Fox.* This is used for raising material from open cuts below the level of the surface; also for stacking ore or fuel on the surface. It consists of a rope stretched between two masts which may be approximately level or on an incline. Running on the cable is a frame on which are mounted four sheaves, known as a bicycle or carrier. The two upper sheaves rest on the cable; the two lower are used for raising and lowering a skip vertically. A stop attached to a cord is placed on the cable where it is desired the bicycle shall remain while the skip is being lowered. When the skip is fully raised, by continuing the pull the bicycle is drawn to the place where the skip is to be emptied.

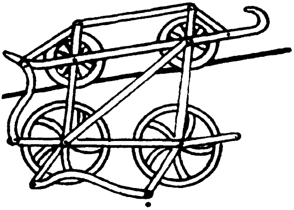


FIG. 61. THE BICYCLE

EXPLOSIVES

Explosives suitable for mining operations should have a destructive action as distinguished from the propelling action required for artillery. Mine explosives should be stable under ordinary circumstances, and not be easily detonated by ordinary mechanical shocks.

Blasting powder explodes at a temperature of about 300° F. One pound weight of powder occupies about 28 cu in. A 1 in. diam. hole contains 6 oz/ft. In the U.S.A. the letters CC, C, F, FF, FFF, and FFFF designate the different sizes of grain. Those of CC represent the largest, about ½ in. diameter, and FFFF the smallest, about $\frac{1}{16}$ in. diameter. Blasting powder is usually exploded by means of a safety fuse. The detonating rate of blasting powder is 1538 ft/sec; while nitroglycerine has a detonating rate of 23,600 ft/sec. The relative volumes of gases given off are 390 from blasting powder, and 747 from nitroglycerine. Low explosives such as blasting powder have a slow rending action, and are fired by direct ignition; while high explosives, such as nitroglycerine compounds, have a quick shattering action and are exploded by detonators. The effect of low explosives has been compared to a push, and that of high explosives to a blow. There is no sharp line between low and high explosives: different kinds merge one into the other.

Nitroglycerine is only soluble in water to the extent of 0·003 per cent. It freezes at 46·4° F., but with the addition of 20 per cent glycol, it has the freezing point lowered to zero or below. When frozen, nitroglycerine compounds are very sensitive. They should be thawed gradually and never over a direct fire or at a temperature higher than 122° F. It should be stored between the temperatures of 50 and 350° F. as below and above these temperatures it is dangerous to handle. On a small scale, dynamite may be thawed in a thawing box, which is a water-jacketed vessel the temperature of which is not more than the hand can bear (about 130° F.). The cartridges should be laid out on slats so that the hot air can circulate among them. On a larger scale the magazine is heated with hot water pipes. Small quantities of dynamite will burn without explosion at a temperature of 356° F. but at 423° F. it explodes violently. In large quantities, it will explode at 356° F.

Gelignite is a little stronger than No. 1 dynamite, and is safer, being less sensitive to shock. It is slower in action and has a less shattering effect.

Gelatine dynamite is about 25 per cent stronger than gelignite.

Blasting gelatine is 40 per cent stronger than gelignite and is used for very hard rocks. It is suitable for tropical climates as it does not leak.

Ammon-gelignite is cheaper than ordinary gelignite with which it ranks in power, but it is hygroscopic. This objection is overcome by dipping the cartridges in a special wax solution.

There are certain materials harmless in themselves, but when brought together are dangerous; such as rack-a-rock, and L.O.X. The latter consists of powdered carbon soaked in liquid oxygen. It does the same work as 40 per cent gelatine dynamite. When fired with a fuse it acts as a low explosive, but when detonated it acts as a high explosive. It is not much used for general mining purposes. It must be fired within 5 or 6 min of charging or it will give off more CO than dynamite. As the cartridges are highly inflammable, they should not be used where there are naked lights.

There are numerous permitted explosives for use in fiery or dusty coal mines. The chief requirement is that they shall not produce a flame which might cause an explosion. All explosives, if used in excessive quantities, will ignite firedamp or dust. About 1 sec is required to ignite mine gas at a temperature of 1000° F. If the amount of heat is less, the time required for ignition is longer. With permissible explosives, the temperature produced should not be over 600 calories per kg and the duration of the flame is about $\frac{1}{35}$ sec, so there is a charge limit of 500 to 1000 g according to the nature of the explosive.

More explosives are required underground to do the same amount of work than on the surface, as there are more tight corners. Certain explosives can be used in open cuts which are unsuitable for use in a confined place, on account of harmful gases given off. It is generally found advisable to use two or more grades of explosives in a mine according to the class of rock to be broken.

It is sometimes cheaper to use more, or a stronger explosive than is actually required to break the ore, such as when it is difficult to maintain the supply of ore; when the labour available is insufficient or inferior; or when there are not sufficient machines.

Deep drill holes from 10 to 25 ft have to be chambered (bulled or sprung) to hold sufficient explosive to do the work required. At first a small amount of high explosive is employed to shatter the rock. To determine the length at the bottom of the hole it is desired to chamber, generally about 2 ft, a stemmer is used. Cartridges are dropped down till the stemmer shows the length has been filled. Tamping is rarely used. The depth in the hole to be sprung generally depends on the height of the face, the number of free faces, and the position of planes of weakness. The chambering of the hole may have to be repeated two or three times with heavier charges, but after the first explosion blasting powder is used.

Before springing a second time certain precautions should be taken. The hole must be sufficiently cool; there should be no smouldering paper left from the cartridges, or tape from a safety fuse. The time limit of half an hour is not satisfactory, as the factor is temperature, not time. Water may be poured down to cool the rock and extinguish anything that is smouldering. If there are no leaks in the rock, the quantity of water required to fill the space up to the top of the chamber will give an idea of the space available for the explosive. When the hole is finally charged, it is tightly tamped. By chambering, the holes can be placed farther apart, thus saving cost of drilling. Some miners test the temperature of a chambered hole by leaving a metal rod in it for a certain time, and then testing it with a hand; but here the personal equation comes in, and anyhow it does not give notice of smouldering paper. If the temperature is to be taken, it is better to use a thermometer with a maximum indicator, and not to recharge till the temperature is reduced to 80° F.

Giant Blasts. It is sometimes desirable to bring down several thousand tons of rock in one blast. An entrance is driven into the face of the rock, about 3 by 4 ft, the distance depending on the height of the face and strength of the rock. At the end of this branches are driven parallel to the face. The powder drifts are widened out sufficiently to hold the necessary amount of explosive. If the ground is wet, precautions must be taken to keep the explosives dry. An estimated charge of polar forcite is 0·25 to 0·40 lb/ton of overburden. Giant blasts are more under control than chambered holes. The stone is seldom projected for a distance greater than 1½ times the height of the face attacked. The charge is fired by electricity, but a time fuse is also laid in case anything goes wrong with the electric fuse.

Fuses. Safety, or time, fuse may be single or double tape. For mining purposes double tape is the better. It should burn at the rate of 2 to 3 ft/min. A special fuse is made for use in collieries where there is fire-damp, which is guaranteed to burn without giving off flame or sparks laterally. Volley firing can be used where there is no electrical equipment. The volley fire has several instantaneous fuses connected to it, which burn at the rate of about 500 ft/sec. Fuses are best capped in some clean place away from the face where a detonator might become lost among broken stone, and cause an accident.

Cordeau consists of a small leaden tube filled with trinitrotoluene (T.N.T.), which has a velocity of 17,000 to 23,000 ft/sec. It cannot be exploded by hammering, pinching, or burning. In use it is exploded by blasting or dynamite. Sold on spools of 100 to 500 ft, but not all in one length. Mostly used for deep jumper holes. On account of its weight and liability of the lead casing to be cut by sharp pieces in the tamping, it is largely replaced by Cortex, which is P.L.N.T. encased in a textile waterproof covering. This is lighter, more flexible, and cheaper than Cordeau.

Electric firing is used in places from which it is difficult to escape; also in coal mines liable to fire-damp and coal dust. The great advantage in the use of electric firing is the absence of danger from hang fires, also men have unlimited time in which to retire before firing. There are two classes of electric fuses, high and low tension. They are fired by small hand dynamos known as exploders. High tension fuses are not much used in mining as they are more liable to deteriorate and cannot be tested beforehand with a galvanometer as a low tension fuse can. An electric detonator is generally completely buried in the explosive of the primer to prevent it from being accidentally withdrawn. High tension fuses are connected in parallel. Low tension fuses are always connected in series.

Detonators

Many explosives require a heavy blow to explode them. This blow is generally delivered by a detonator. The detonator used should be strong enough to generate the full force of the explosive. It is false economy to use too weak a detonator, and there is no advantage in exploding blasting powder with a detonator since it explodes just as well with a spark from a fuse.

Ordinary detonators are graded according to the amount of fulminate of mercury contained in a copper cap, which varies in length accordingly. In England they must be graded as follows—

No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7	No. 8	Fulminate
4·6	6 2	8·3	10	12·3	15·4	23·1	30·9	Grains
0·3	0·4	0·54	0·65	0·8	1·0	1·5	2·0	Grammes
¾	⅞	1	1⅛	1¼	1⅜	2·0	—	Length of cap in in.

In the United States, the strength is expressed as x, xx, xxx, xxxx, xxxxx. One objection to the fulminate detonator is its tendency to absorb moisture. Lead azide detonators put up in aluminium caps overcome this difficulty and are also safer to handle.

Magazines

Explosives must be kept in a dry place away from other buildings. The ground should be cleared for at least 50 ft on every side in case of bush fires, and should be enclosed by a fence not easy to climb, say 10 ft high. If there are more than one magazine, they should be a suitable distance apart, and earthen mounds, the height of the magazine walls, made between them. The material of which the magazine is built depends on whether danger is expected from within or without. If from within, it should be constructed of light material, such as wood or galvanized iron, which, when blown to pieces, would not be thrown a great distance. If from without, as in case of a stray bullet, bush fire, or action of malicious persons, the magazine should be built of stone, brick or concrete, or excavated in solid rock or in a mullock heap. If built of stone or brick and the magazine gets blown up, the fragments fly a considerable distance; but if constructed of a lean concrete made of six parts of sand to one of cement, and protected by 1 in. boards spaced about 4 in. apart, it not only offers an effective resistance against bullets, but in case of an explosion breaks up into small pieces.

A magazine should be built 2 ft above the ground on brick, stone or concrete walls and piers with a dampcourse of ruberoid. Iron plates are placed on these, on the top of which is a 3 in. concrete floor. The walls are made double, a cavity being left for the sake of ventilation, with ventilators suitably protected at the bottom on the outside, and at the top on the inside. The roof should be made double with sand between, so that if anything heavy should be dropped on the roof and break it, it is less likely to penetrate the sand. A lightning conductor should be placed at each end of the building. The inside, ceiling, sides and floor are all lined with wood. No iron must be exposed inside the building. There are two doors opening outward at right angles to one another. The porch between acts as an air-lock in cold weather. In cold climates the dynamite must be kept thawed by means of hot-water pipes at a temperature between 70 and 100° F. The water is heated in a special building some distance away from the magazine. Detonators should be kept in a separate building. Safety fuses can be stored in any dry place. The magazine should be in charge of one man only. Explosives should be used up in the order in which they are received. Cases containing high explosives should be opened with hardwood wedges and mallet. A limited amount of explosives are kept in an underground magazine under lock and key, for distribution to different parties. These parties have strong boxes in which to keep their explosives. Only electric or an enclosed light should be allowed in a magazine.

Blown-out and Windy Shots. These are not the same, though the result may be similar. An exploded charge may not be strong enough to do the required work and may result in a blown-out shot when the tamping is blown out; or it may be due to the placing of a hole so that it cannot break the rock; or the tamping may be insufficient to resist the force of the explosion. A windy shot results when a portion of the gases escape through the tamping or some crevice in the rock before completing its work of rupturing the rock.

Hang-fire or Misfire. A charged hole, unless fired by electricity, may mis- or hang-fire. It is not possible at first to tell which it is; so it is advisable to assume it is a hang-fire, and act accordingly. One should, of course, count the shots as they go off, but if two or more explode simultaneously, the count may be confused. A hang-fire with a safety fuse may be due to a break in the powder core, when there is always danger of a spark smouldering and slowly creeping past the obstruction, so men should not return to the face for an hour and a half after the fuse was lighted. Old dynamite is liable to burn without exploding. In wet holes, if the cartridges are not pressed down well, water or sludge may come between them and prevent the whole charge from exploding, and the noise of the primer going off may deceive one. Water may enter the detonator and prevent it exploding. If the fuse is not cut square, the fibres of the outer covering may cover the powder core. Some sawdust may have been left in the detonator which prevents the spark from reaching the fulminate. If blasting powder has been used, it can be made harmless by carefully withdrawing the fuse and pouring water down the hole to dissolve the nitre.

Though there is no danger of a hang-fire with electric fuses, yet it is possible there may be a misfire if the exploder is over-loaded or the wires and fuses are unsuited for one another. Detonators of different kinds should never be used in the same round of blasts. Joints should be well made, and there must be no short-circuiting. In case of misfiring, the charge should not be withdrawn, but the tamping can be removed to not nearer than 6 in. or farther than 12 in. from the charge, as determined by a tamping rod, and a fresh detonator and primer placed and exploded on the top. Sometimes a wad of paper is placed between the charge and the tamping, which acts as an indicator when it is time to stop extracting the tamping.

Never drill a hole towards an unexploded hole with the object of exploding it; but drill a hole parallel to the misfire sufficiently far away to be level with the top of the charge, with the object of exploding it.

Destroying Explosives

It is sometimes necessary to destroy explosives, generally because they have become dangerous or unreliable.

Black powder can be rendered harmless by placing it in water to dissolve out the saltpetre, which is part of its composition.

Greenish spots on the casing of nitroglycerine explosives are an indication that nitroglycerine has decomposed and in consequence is very dangerous. If only a small quantity has to be destroyed, it is best done by detonating the plugs with a primer of fresh explosive. An 8 oz primer can be used for every 3 to 5 lb of old explosive in order to ensure complete detonation. The whole should be placed in a pit and covered up with a little fine earth.

When the quantity to be destroyed is fairly large it is best done by burning (except for samsonite No. 3) on a suitable plot of waste ground. Some sawdust is laid on the ground about 8 in. wide and 1 to 2 in. deep. The cartridges are partly unwrapped and laid down in the sawdust arranged in herring-bone fashion so that the V-shape, formed by the cartridges, points in the direction of the wind which is blowing. A paper train is then laid on the leeward side of the explosive so that ignition takes place against the wind which reduces the risk of explosion due to too rapid burning. Not more than 25 lb of explosive should be burnt at one time. The explosive having been laid out as mentioned above, kerosene is sprayed over the whole to ensure complete burning, and the paper train lit. The operator should retire to a safe distance while the explosive is burning in case there should be an explosion. When the fire has completely died out, a search for undestroyed cartridges should be made.

Safety Precautions

No boy under 21 should be allowed to handle explosives.

No drill hole should be bored in any remaining portion of a hole in which a charge has previously exploded.

No more holes should be charged in any one face than are intended to be fired in one round.

Persons intending to fire should give definite warning to all persons in the vicinity who might be injured by the blast by calling out "Fire" twice, and all places of ingress to where the blasting is taking place should be guarded, either by a person or by a notice.

The shot-firer when igniting a number of holes in a round must allow ample time in case of unexpected delay through difficulty in lighting a fuse or other cause. When more than one shot is exploded at a time, count the number as they go off to ascertain if they have all exploded or if there is a mis- or hang-fire. Sometimes two or more shots may go off simultaneously, in which case the only guide is the louder sound.

No person should enter any place after firing until the fumes arising from the explosion have been sufficiently dispersed and the place has been examined by a competent person.

To prepare the primer when using a safety fuse, cut the fuse squarely across with a sharp knife. Shake the sawdust out of the detonator. If it cannot be removed by gently tapping against the palm of the hand, discard it as unsafe for use; never attempt to scrape it out. Never let the ends of the fuse get damp or gritty. Slip the detonator over the end of the fuse till the composition and fuse come in contact. Never screw the fuse against the composition. Crimp the detonator gently but firmly on to the fuse with special fuse pliers; never bite it on. Open the cartridge, make a hole in it parallel to its length, and place the detonator in the hole. If in a wet place make the joint between the detonator and fuse perfectly water-tight with grease. Tie the paper of the cartridge round the fuse. There are three forms of detonators: (1) the ordinary; (2) the electric; (3) the delayed.

No explosive should be forcibly pressed into a hole. The borehole must have a clearance of at least $\frac{1}{8}$th of an inch over the diameter of the cartridge.

Each charge should consist of not more than one description of explosive.

No iron or steel tool should be used when charging or ramming; and where any explosive containing nitroglycerine is used only wooden tamping bars should be employed, and the charge should be pressed home, not in any case rammed.

Clay or earth free from stones should be used for stemming, and should extend from the explosive to the collar of the hole.

Tamping for the first 6 in. should be done by gentle pressure. Pressing hard should not be done till there is a good cushion of tamping material above the charge.

An accurate record should be kept by every person charging a hole of the distance between the top of the charge and the collar of the hole, and a mark should be made showing the direction in which the hole has been bored.

A charge that has missed fire should not be drawn, but the tamping may be carefully removed to a depth distant not more than 12, nor less than 8 in. from the outer end of the charge, and a fresh charge inserted for the purpose of exploding the original charge.

In coal mines before any explosive is charged in a hole all dust and drillings must be removed from the hole.

Shot-firers must not fire a shot in any place until the roof, ribs, and floor have been thoroughly washed down between the face and the rock-dusted area.

Coal dust should never be used for tamping.

No more than one shot should be fired at a time.

Shot-firers must examine places for gas before and after each shot is fired.

None but experienced shot-firers should be allowed to handle detonators or charge or fire shots.

When a shot is fired by electricity, the shot-firer should use at least 100 ft of leading cable and must be out of direct line of fire.

In the event of a misfire when safety fuse is used, as it may be a hang-fire, no person should return nearer than a safe distance until the expiration of an hour and a half from the time of lighting the fuse. If there is a misfire when electricity is used, the cable should be disconnected from the firing apparatus, and the hole should not be approached for at least ten minutes, except when the misfire is due to faulty electrical connection when the shot is fired as soon as possible after the defect has been remedied. In other cases of misfires, another hole should be drilled at least 12 in. from the unexploded hole and as near as possible parallel to it, charged and fired. Before firing the second shot, a string should connect the fuse of the first hole to a prop or other stable object. After the second shot has been fired no one should be allowed at the face till the person who has fired the shot or an official has made a careful search for the detonator and charge. If these are not found at the face, the skip containing the blown-out material should be marked for special examination at the surface.

The lead wires of electric fuses should be handled with care to avoid being kinked or twisted. Care should be taken to prevent the leading cable coming into contact with rails, pipe lines, or skips when shots are being fired.

When testing individual low-tension electric detonators, the detonator should be shielded in some way, as in an inverted iron pot or a pipe. The bared ends of the wires are scraped clean and applied to the terminals of a galvanometer. If the needle is deflected the conductivity of the electric detonator is satisfactory. If the needle does not move, there is a fault somewhere.

Electric exploders must be sufficiently powerful for the work they are called on to do.

Misfires may be due to a variety of reasons. If a safety fuse is used the core may be pinched owing to bad tamping.

If the fuse has not been cut square, a flap may get between the core of the fuse and the compound of the detonator. The detonator or explosive may have deteriorated owing to prolonged storage, the presence of sawdust left in the detonator, poor cramping, etc.

A bulled or chambered hole should be thoroughly washed out with water to extinguish any smouldering material that may be present before charging again. It should not be recharged within four hours of the bulling to give the rock time to cool sufficiently, or until the shift boss or other person in charge of the work has examined the hole and satisfied himself that the new charge can be inserted with safety.

All explosives must be kept in the custody of the manager of the mine or some responsible person appointed by him: no other person should be allowed access to any magazine.

No matches, iron, steel, grit, or anything that might cause a spark should be introduced into a magazine. Any light taken into a magazine should be enclosed.

The man in charge of a magazine should refuse to supply explosives to any unauthorized person or to any one who has not a proper case or canister.

No explosive should be conveyed about except in covered receptacles. Old stock explosive should be used up first.

Explosives must not be stored underground in any quantity exceeding what is likely to be required for use during six working days for the purpose of the mine.

Explosives when taken from the underground magazine should not be taken for use into the workings unless in a securely covered case or canister, each of which should contain not more than 16 lb blasting powder or 10 lb high explosive. Detonators should be carried into the mine in a separate leather or other suitable container which must be locked before being taken from the magazine and remain locked till required for use.

A workman should not have in his possession more explosive than is required for the shift on which he is engaged, and except when actually in use for the purpose of charging explosives it should be kept in explosive boxes in a safe place.

All unused explosives and detonators must be returned to the underground magazine at the close of each shift, and the number of detonators checked against the number of holes fired.

Any excess of explosives in underground magazines at the end of the week should be returned to the main magazine at the surface.

There are three kinds of magazines for explosives: (1) The main storage magazine on the surface. (2) Underground service magazines. (3) Magazine boxes where parties keep the explosives they are likely to require during a shift. These are capable of being removed from time to time as required.

The size and design of a surface magazine depends on local requirements. The material of which it is built (wood, brick, stone, or concrete) is often determined by what is available. It may even be excavated in solid rock or a mullock heap. If constructed of lean cement mortar (6 pt. sand to 1 pt. cement) protected by 6 in. boards 1 in. thick, spaced about 4 in. apart, it is not only effective against bullets from outside, but in case of an explosion it will break up into dust, thus reducing damage to people and property in the neighbourhood to a minimum.

A surface magazine should be erected in a dry place a reasonable distance from other buildings and paths. It should be fenced with a wall about 10 ft high and kept clear of undergrowth for at least 50 ft all round in case of bush fires. It should be constructed so as to prevent unlawful entry or the introduction of fire. It should be well ventilated and protected from extreme heat or cold. A temperature of 65 to 70° F. is an ideal temperature for high explosives when being used. Though nitroglycerine freezes at 45° F., it is liable to become chilled and hard before this temperature is reached. There are now some grades of high explosives which freeze at a lower temperature, but they are somewhat more hygroscopic.

The bottom of a built-up magazine should be about 2 ft above the ground and rest on piers and outer walls. The walls should have galvanized iron ventilators let into them. Steel plates should rest on the walls and piers, and on these is a 3 in. layer of concrete. Windows should be dispensed with when possible, but if absolutely necessary they should be provided with wired glass and protected with double shutters capable of being opened from the inside only. The walls are made double so that air can enter from the bottom. The walls and ceiling are lined with tongued and grooved boards. The floor is covered with soft-wood flooring boards; or the cement is covered with 7 lb sheet lead, burnt at the joints and turned up 4 in. all round the walls. There must be no exposed iron or steel inside the building. If iron nails are used they must be counter-sunk and puttied. Locks and hinges should be of brass or gunmetal. Keyholes should be closed on the inside. There should be two closely-fitting doors arranged at right angles to each other, both opening outward. The inner door should be of wood; the outer door of wood faced with iron on the outside secured with galvanized iron bolts and nuts, the nuts being on the inside. A sliding board about 6 in. high is placed just inside the magazine as a reminder that no one should go past this without wearing overshoes of an approved pattern provided. The board is easily removed when necessary to sweep out the magazine. The building should be provided with one or more lightning conductors of round copper or iron rod raised at least 4 ft above the roof, with three terminals, earthed by copper tape connected to a sheet of copper embedded in coke or other loose material, 12 ft away from the magazine and kept damp. The conductor should be tested annually for conductivity, and occasionally for resistance, which should not exceed 15 ohms.

An underground magazine should be in a dry place, lined with soft wood, tongued and grooved, in a position approved by an inspector of mines. It should be separated from working places in the mine by a substantial door kept locked when not in use. The chamber is provided with benches.

Explosives become defective in course of time, and have to be destroyed. In the case of blasting powder, it is sufficient to place it in water, preferably hot, in order to dissolve the saltpetre, one of its constituents, when it becomes harmless. To destroy nitroglycerine compounds, remove the cartridges from their packages and lay them out in a train on a piece of waste land at an angle of 45 deg. to the direction of the wind. Soak them well with kerosene and ignite the weather end with a length of safety fuse or other means to give the operator time to reach shelter. Detonators when possible should be taken out to sea and dumped; they should never be thrown into a pond or water course. If this cannot be done, fire them in batches in some suitable place. Fuses should not be capped at the face in case a detonator is dropped among loose material and cannot be found.

Nitroglycerine when frozen is very sensitive and should be thawed gradually. Small quantities may be warmed in a jacketed vessel. The water in the jacket should be as hot as the hand can bear, say 90° F., never above 122° F. As the water cools fresh water of the necessary temperature is added. The vessel should never be heated direct over a fire, neither should the cartridges be placed in a stove or in hot water. Cartridges are built up in horizontal layers, alternate layers being arranged at right angles to those above and below. If a large quantity is to be thawed, it should be done in a separate building warmed by hot water pipes, or radiator; never by an open fire. Dynamite should be stored at a temperature between 50 and 104° F.

ACCIDENTS

Bleeding

This may be external or internal; arterial, venous, or capillary. Arterial blood is bright red, and at first escapes in spurts. Venous blood is purplish. Capillary blood is lighter than that of venous blood. Arteries are deeper seated than veins, and there is an artery corresponding to each bone in a limb. The injured artery should be pressed against a bone between the wound and the heart; the side against which you press must be supported. Digital pressure should be maintained until more permanent means can be employed. If necessary for one man to take the place of another who is using digital pressure, he should place his thumb on that of the other so that when the latter is removed there will be little loss of blood. If bleeding is from a limb raise it. Remove any foreign body from the wound that can be seen, wash away as much dirt as possible with clean water, and cover with a clean dry dressing, such as a pocket handkerchief or even unprinted paper, and tie firmly with a bandage. Fig. 62 shows the position of the main pressure points.

In cases of internal haemorrhage, loosen any clothing about the neck; sprinkle cold water on patient's face and over the region of the haemorrhage, if known, and fan. Blood from the lungs is bright red, is coughed up and frothy in appearance. Blood from the stomach is dark red, vomited and may be mixed with food.

Fractures

Signs of fractures are pain at or near the point of fracture; inability to use the part below the break, swelling, the limb assumes an unnatural position; the limb may have unnatural mobility; if the two ends of the bone rub against one another it causes crepitus. Neither unnatural mobility nor crepitus should be deliberately tested.

If the fracture is accompanied by bleeding, stop the bleeding first as being more important than the broken bone.

In the case of a compound fracture, dress the wound before applying the splint. Treat doubtful cases as fractures. Do not apply a splint over a wound if it can be avoided. See that the bone is carefully pulled into place. Apply the splint before removing the patient if possible, and be careful not to

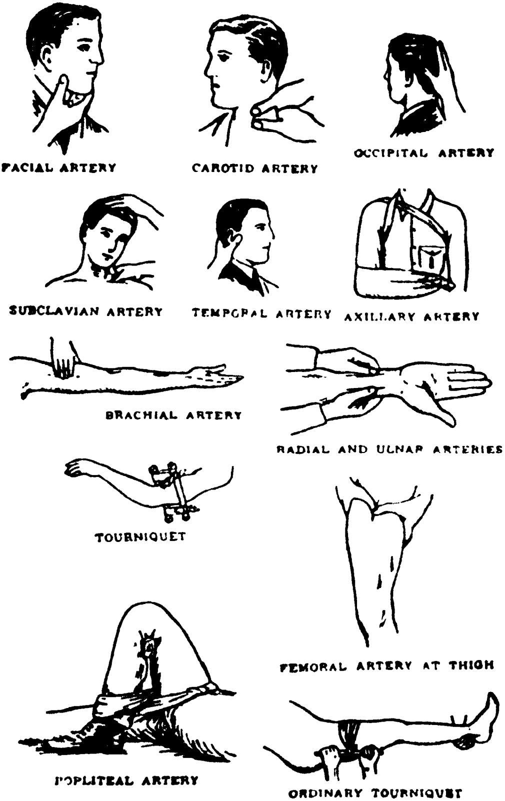


FIG. 62. PRESSURE POINTS

convert a simple fracture into a compound one. First attach the splints to the sound part of the limb. Do not apply a bandage over the break itself. All bandages are tied with a reef knot which should be on the outside splint. Splints can

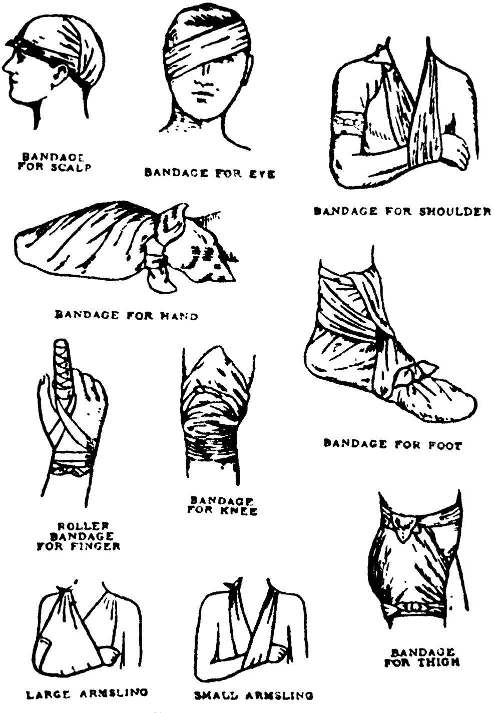


FIG. 63. BANDAGES

be made of various articles depending on what is available, and if possible place padding between the limb and the splint. If nothing else is available, a broken leg may be tied to the sound one; or a broken arm fastened to the body. Bandages must be firmly tied (Fig. 63).

Useful lengths and widths of splints are as follows: Hand, 10 in. by 3 in.; Foot, 13 in. by 3 in.; Arm, 15 in. by 3 in.; Forearm, 19 in. by 3 in.; Knee cap, 36 in. by 4 in.; Leg, 30 in. by 4 in.; Thigh, 58 in. by 4 in.

Bandages

Bandages may be applied to keep a dressing on a wound; to support some part of the body, such as an arm; or to hold splints in place. Examples are shown on Fig. 64. There are two kinds of bandages, triangular and roller. The latter vary in width according to the part of the body to which they are applied. For fingers the bandage is ¾ in. wide; for hand and wrist or eye, 2 in.; leg or forearm, 2 to 3 in.; head, 2¼ in.; arm, ear, elbow or breast, 3 in.; knee, 3½ in.; groin or shoulder, 4 in.; chin with four tails, 6 in.

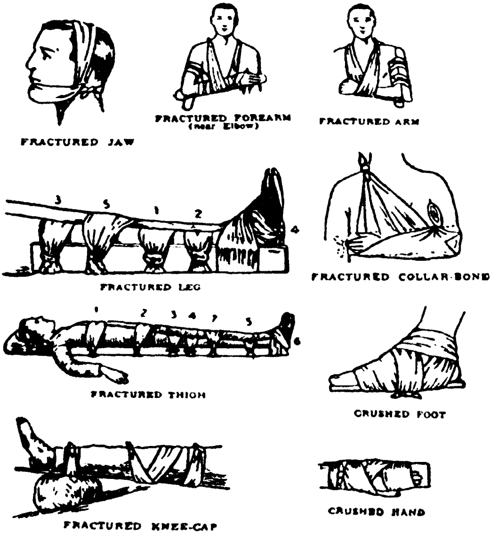


FIG. 64. APPLYING BANDAGES

Triangular bandages are made out of calico 36 to 40 in. wide cut into squares and then divided diagonally across. To apply a large arm sling, place the bandage over the front of the patient's body so that the point of the bandage is towards the elbow of the arm to be supported. The upper end of the base is passed over the shoulder of the sound side and allowed to hang over the shoulder of the injured side, except in the case of a broken collar bone; the lower end of the bandage is now brought up outside the arm and the two ends tied together in a reef knot so that it occupies the hollow near the neck. The point of the bandage is now brought round the elbow and fastened to the sling with a safety pin. The sling is so adjusted that the little finger is slightly above the level of the elbow.

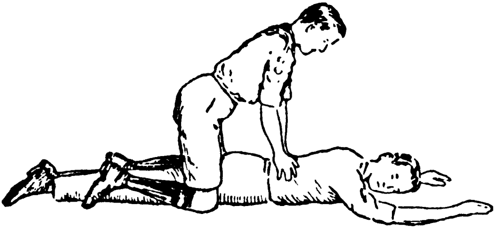


FIG. 65. RESUSCITATION

If there is no bandage available, the bottom of the patient's coat or shirt may be turned up and fastened higher up. The small armsling, which is narrower, is adjusted in a similar way.

Schafer's Method of Artificial Respiration

Used for restoring suspended animation in cases of drowning, suffocation by gas, or severe electric shock. Do not waste any time in starting operations such as by loosening the patient's clothing, but feel in his mouth for any foreign body such as false teeth or tobacco, and remove them. Lay the patient face down. Straighten the body. Place one of his hands under his forehead and the other straight forward in line with his body (Fig. 65). Do not put a pad under his abdomen with the idea of draining water out of the patient's body in the case of drowning, as this would lessen the intake of air by the weight of the body continually pressing on it, thus constricting the air passages.

Kneel down straddling the patient's thighs and face toward his head. Place a hand, one each side of the small of his back as close to his hips as possible, the thumb and fingers extending over the lower ribs. Throw the weight of your body forward on your arms which are kept straight. By this means the air and water are driven out of the patient's lungs. This forward motion should occupy a period of three seconds.

Then raise your body slowly for two seconds removing the pressure formerly exerted and allowing air to enter the lungs. Keep your hands in the same position all the time, do not remove them from the sides of the patient. The movements must be strong and regular, not violent or carried out in a jerky manner, and there must be no marked pause in the cycle. Artificial respiration should be carried out for at least half an hour. Patients have recovered after 3½ hours.

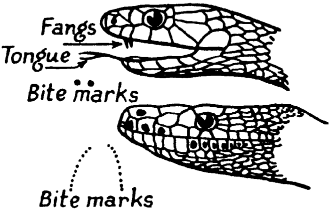
Electric Shock

Remove the victim from the live wire if he is still in contact with it. Remember moisture—perspiration, rain, etc.—conducts electricity; so do metals—e.g. nails in boots—so be careful how you handle the victim. Use a dry coat, dry rope, dry crooked stick, or any other non-conductor. If you have to touch his body, cover your hands with some dry non-conductor, stand on a dry board or some insulating material. A line may be short circuited by throwing a piece of iron wire or an iron bar across the two conductors of the circuit; but care must be taken to let go before the contact takes place. If the patient has ceased breathing, apply artificial respiration.

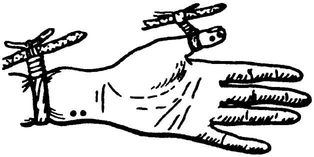
Snake Bite

The forked tongue of a snake is sometimes mistaken for a sting. A snake does not sting; it bites or strikes. A venomous snake is easily distinguished from a non-venomous snake by noticing the smaller number of scales along the jaws. As a rule, it is only the fangs that show any mark when a poisonous snake strikes. These fangs, one on each side of the upper jaw, lie flat when not about to strike. Since the bite of a venomous snake has to be treated at once, it is better to kill all snakes on sight and examine them afterwards.

Tie a ligature close to the bite between it and the heart so as to localize the effect of the poison. The ligature may consist of a piece of string, a boot-lace, or a strip of a handkerchief. Make a loop, insert one end of a stick in it and twist round very tight. Tie the longer portion of the stick to the limb to prevent it unwinding. Wipe away any venom that may be on the surface of the skin. Make a longitudinal cut over the fang punctures as deep or deeper than the punctures, so as to drain away the poisoned blood. Suck or squeeze out as much of the blood as possible. Do not make the incision across from one puncture to the other. The cut can be made with a clean knife, piece of glass, etc. Force crystals of permanganate of potash into the cuts to neutralize the poison. The ligature should be loosened in half an hour's time, otherwise mortification may set in. Retie and loosen the ligature every five minutes for another half hour. All poison snakes are not deadly.



Poisonous and non-poisonous



Correct application of Ligatures

FIG. 66. BITES

Occupational Ailments

Though men are liable to all sorts of diseases, those working at certain occupations are more subject to some than others. Prevention is better than cure, and many of these troubles can be avoided by taking certain precautions. The management may provide the means, such as suitable lighting, ventilation, drainage, underground and surface latrines, drinking water, properly appointed change house with hot and cold showers and means for drying wet clothes, formulate safety rules, and have a first aid outfit and someone who knows how to use it. The men on their part should carry out instructions and use their common sense. Men susceptible to certain diseases should not be employed where they are likely to contract them. Sickness is not only a loss to the patient but also to the mine. A sick man, even with a minor injury, cannot do a fair day's work.

Silicosis. This is one of the most serious diseases to which miners are liable. It comes on gradually which causes men to look on temporary inconveniences of more importance.

Silicosis is due to fine particles of dust of certain minerals such as quartz and sericite which lodge in the lungs and prevent them functioning properly. The coarser particles are filtered off naturally. Anthracosis, the coal miner's equivalent, is not nearly so dangerous, and often wears off when the patient ceases to work where there is coal dust for employment in fresh air. In the case of silicosis, the lungs being weakened are more liable to tuberculosis. The times required for particles of different sizes to fall a distance of 7 ft in still air are given as

1 micron	5 h 30 min
2 microns	1 h 25 min
3 microns	33 min
4 microns	19 min
5 microns	12 min
10 microns	3 min

If the air is in motion, of course, the particles remain longer in suspension, but if they are weighted with water from a spray they will sink more quickly. The chief sources of the dust are drilling, blasting, and handling. When drilling by hand, a wet swab should be kept round the drill at the collar of the hole. Before blowing out a hole preparatory to charging it, wet it thoroughly. As a rule, blasting should only take place at the end of a shift. A water spray should be started before the fuse is ignited and kept on for 30 min after the blast. If driving a heading, a damp screen can be set up at a suitable distance from the face, and the walls sprayed. The use of water in a hot mine is objected to by miners though it is less harmful than the dust. Men should not return to the face after blasting for a period of at least 30 min. Respirators are not suitable for miners at work as they soon get clogged up with dust. Suitable ventilators suck out the fine dust. When possible, the men should always travel in the downcast shaft with the fresh air. Dust is also stirred up when shovelling and when walking in dust. Dusty roadways should be thoroughly wetted. It is no good just to moisten the surface, for the feet sink through to the dust beneath and stir it up. Breathing in aluminium dust has proved useful for the prevention and cure of silicosis in many cases.

Powder Smoke Headache. Men are liable to this when they first commence mining. The symptoms are intense headache, twitching of the muscles, pupils contract, eyes become blood-shot, and vomiting. Men get accustomed to it in time, and it only troubles them again if they get more powder smoke than usual.

Lead Poisoning. This affects men, more especially those working in lead carbonate stopes. Miners breathe in the dust, do not rinse out their mouths before taking a meal, and perhaps do not wash their hands.

Salivation. This is a trouble in quicksilver mines, but only where metallic mercury is present. Where this is the case, spray with lime-sulphur which reacts on the mercury. The symptoms are excessive flow of saliva, diarrhoea, haemorrhage, vomiting, dizziness, and headache.

Pneumonia. Brought about by a sudden change from the hot atmosphere of a mine to the cold surface air and wind. There should be a covered passage from the shaft to the change house.

Chronic Bronchitis. Most miners are mouth breathers, so a rapid change from warm workings to a cool shaft, together with changes in the atmospheric pressure, breathing dust and smoke-laden air, produces chronic catarrhal conditions. This is not in itself dangerous and can frequently be cleared up by rest and outdoor life, but by lowering the vitality of the mucous membrane makes it a suitable nidus for tubercle bacilli.

Chronic Laryngitis. This is due to the peculiar way miners expel air from their lungs when using a hammer, which irritates the vocal membrane.

Carbon Monoxide. This gas is lighter than air, so is found near the roof. It is formed by the incomplete combustion of organic matter, as during a fire. It will burn in air, and explodes when mixed with air in certain proportions. It is very poisonous and is quickly absorbed by the blood in which it accumulates; 0·06 per cent in the air will cause the blood to accumulate 30 per cent after an hour and a half which causes giddiness, palpitation and shortness of breath; 10 per cent in the air causes helplessness in about an hour during which time it accumulates in the blood to the extent of 50 per cent, and causes loss of power over the legs; the accumulation of 80 per cent causes death. A man may die several days after having been poisoned by this gas.

Carbon Dioxide. This gas causes suffocation by taking the place of oxygen. Being heavier than air, it collects in low places. The gas may be stirred up and mixed with air if in a shaft or well by attaching a bag to a rope and moving it up and down. Men can work in an atmosphere containing 3 per cent of CO_2, but quickly become fatigued, and much exertion will cause panting. At 4 per cent breathing becomes distinctly deeper and the rate of breathing increases as if the man were exerting himself. At 5 per cent there is marked panting when at rest. With 7 to 8 per cent the panting becomes more and more distressing, the face becomes flushed, the pulse stronger and more frequent, and headache is produced. At 10 per cent the gas begins to have a stupefying effect, giddiness, loss of muscular power, profuse perspiration, nausea, singing in the ears, and livid coloration about the eyelids and lips.

Nitrogen. This gas will not support combustion. Sometimes the oxygen of the air has been consumed by minerals, leaving the nitrogen behind in badly ventilated places, which will suffocate anyone remaining there.

Sulphur Dioxide. This is an irritating, irrespirable gas given off by the oxidization of metallic sulphides when heated, and compels men to seek fresh air. Workmen exposed for a length of time to a mild degree of SO_2 fumes suffer from headache, coughing, sneezing, congestion of the chest, smarting of the eyes, etc.

Cyanide Poisoning. People have occasionally drunk cyanide solution out of vats under the impression it was pure water. As the effect of cyanide poisoning is very rapid, where such a misadventure is possible, the antidote should be at hand. This consists of a 25 per cent ferrous sulphate solution, a 5 per cent caustic soda or caustic potash solution, and phials containing 2 g of magnesia in a glass of water. After being retained in the stomach for a few minutes, induce vomiting, or use a stomach pump.

Heat Stroke. The face of the patient is flushed, the pulse quick and bounding, the breathing difficult, the patient develops sickness, giddiness, faintness and thirst. The skin is dry and very hot, and the patient may become insensible. Strip the patient to the waist, sprinkle water on him and fan vigorously.

Heat Cramp. Brought about by drinking large quantities of water in a hot place, causing excessive perspiration which removes too much salt from the blood. To make up for this loss, dissolve a little salt in the drinking water.

Sabmina. Men who suffer from this mostly work in hot places. The sweat glands seem to secrete more sweat than can be carried off, and small blisters are formed.

Nystagmus. This is a constant twitching of the eyeballs, due to working with a flickering light, such as that of a candle in a draughty place.

Ankylostomiasis or Hook Worm. This is spread about through insanitary arrangements underground. A man retiring to a disused excavation to relieve himself, instead of using a proper latrine, may easily tread on excreta of one who is suffering from this disease and when climbing a ladder leave some of the dirt on the rungs. As miners hang on to the rungs when climbing up or down, a miner is liable to get some of the dirt on his hands, and if he does not wash them before eating his crib, he is liable to become infected. The hook worm can be eliminated from places where it infests the ground by sprinkling it well with salt.

Accidents to Men

In most vocations some sort of accident or disease is more common than in others. The cause, means of prevention, and remedies of such conditions should be pointed out to those so engaged in the form of rules and regulations which must be enforced.

In connection with mining, though some rules may apply to all mines, others may be suitable for coal mines, but not to metal mines, or the other way about. Again, some rules suitable for an extensive mine may not be applicable to a small mine, or for one that is being opened up.

Government mine inspectors are allowed a certain amount of latitude in connection with mining regulations.

A competent person is considered to be an officer in charge of that particular work, or a suitable person appointed by him. A competent foreman should inspect working places at least once a shift, and the result of all inspections should be written in a book kept for the purpose and should be attended to by the responsible officer.

If underground workings are extensive, indicators should be so placed that those unfamiliar with the mine can find their way to exits.

Any person observing anything likely to endanger life or property must immediately report it to his superior officer. Reports of any accident should be made immediately and signed by witnesses, as accounts are liable to vary from facts with the lapse of time.

Where more than twelve persons are ordinarily employed at a mine stretchers and first-aid chests with splints and bandages should be kept in a convenient place for immediate use in case of accident.

If more than four persons are employed underground, accommodation should be provided above ground near the principal entrance to the mine (but not in the engine or boiler house) to enable the persons employed on work conveniently to change and dry their clothes. Hot and cold water should be laid on, the temperature of which is capable of being regulated by the user.

At least 10 sq ft should be allowed in the change-house for each man in the largest shift, not counting the area required for showers. There shall not be less than one shower for every five persons, and one basin for every twenty persons in the largest shift.

Suitable sanitary conveniences should be provided both above and below ground. They should be kept in a sanitary condition and in good repair. The roof and sides should be kept lime-washed. Pans must be emptied not less than twice a week. In metal mines, at least one convenience should be provided in each level near the upcast shaft.

All surface openings, whether in use or not, including open cuts, should be securely fenced in. Guards should also be provided to protect winzes, entrances to shafts underground, and other dangerous places.

The batter of an open cut should be at a safe angle, depending on the nature of the material.

Men working on the face of an open cut should be provided with a life line not less than 1 in. in diameter securely fastened at the top, to which should be attached a lighter rope not less than ½ in. in diameter connected at the other end to a substantial belt to be worn round the body of the person.

Falls of Ground. Most accidents in mines are due to falls of ground from the roof of an excavation: not large-scale cavings, but comparatively small pieces of rock or coal from a few pounds weight to a ton or two which might have been barred down or supported by timber. Such accidents are not so spectacular as explosions where several men may lose their lives at one time, but one or two here and one or two there mount up by the end of a year. Accidents due to ground falling might often be avoided if proper precautions were taken. Miners should be educated in safety methods. A sufficient number of experienced shift bosses or deputies should be provided: they should make frequent and thorough inspections, at least once a day. They should consider the character and thickness of overlying strata, their joints and slippery heads. If shale above a coal seam is liable to become winded, a few inches of coal should be left as a protection. As a rule, overlying strata having a high silica content are stronger than others. Strong lights should be used when making an inspection of the roof. Sounding the roof with a hammer is a rough-and-ready method of testing baulky ground, but is not infallible if a loose block is large; hence the necessity of noting the strike and dip of joints and faults.

There are three types of sound that are recognized by miners: (*a*) A sharp ringing note which denotes a normally safe roof. (*b*) A hollow drummy sound indicating a need of caution and support. (*c*) A dull sound often associated with friable or fissile argillaceous material. The barring down of loose material should be done before passing under freshly-blasted ground, to be followed by support when necessary. Timber gives notice of heavy ground by cracking and bending. The geophone has been used to listen for sounds due to rock pressure not otherwise noticed; the idea being that the intensity and frequency of the sub-audible noise would be a good guide to roof conditions.

It is now often possible to tell when rock bursts are likely to occur, and whether or not it is safe to remove supporting pillars that have been left to hold up the roof in open stopes. The geophone with its amplifying and recording equipment is complicated but small, and it can be stuck in an ordinary drill hole. Most rocks under pressure below 1000 ft will emit a noise every few minutes which generally increases with depth. In some cases the rock on both sides of a vein is under strain. If much ground is extracted at a time, there will probably be a rock burst. The only thing to do in such a case is to extract as little ore as possible at a time, and fill the cavity quickly. Should a rock burst take place, the ground should be given time to settle before continuing further extraction. Falls of rock or ore may sometimes be prevented by attacking a block of ground in the proper way. For instance, a wedge-shaped block of ore is loose on the top and bottom and almost so on its walls. It should be attacked from its toe, so as to have a solid roof overhead.

The management and men both have a part to play in preventing accidents: the management by supplying what is needful in the way of materials, frequent and thorough inspections, and the making of suitable regulations. The men by reporting any danger, and carrying out instructions for their safety.

An accident may be minor, such as a cut finger; major, such as the loss of a limb or eye; or fatal, when a man dies within two or three days from the effect of the accident. An accident generally has a bad moral influence on others and disorganizes work. Even a minor accident makes the injured one less efficient.

The various mine accidents are too numerous to mention here, but they are largely due to the carelessness of the victim or his mates. One of the most common is that due to a fall of rock or ore. After blasting, when the atmosphere has cleared, the first thing that should be done is to examine the roof with a bright light, and bar down any loose pieces. The ore or rock left in position should be tapped with a hammer to ascertain by the hollow or drummy sound whether it is baulky. This test, however, is not sufficient if there is a large block which may sound solid and still be dangerous on account of slippery heads. One must be on the look-out for joints, and support any doubtful places. If in a colliery the rock above the coal is shale liable to become windy and fall, it should be protected by leaving two or three inches of coal overhead. Timber gives warning of heavy ground by bending or breaking. The manager should have suitable timber handy for the miner; and the miner should use it at once, not leave it for the next shift. Falls of ground are sometimes caused by not leaving pillars of coal or ore of sufficient strength.

Bursting ground is more likely to take place as the bridge of ore in a stope becomes thinner and weaker. Geophones have been used to detect sounds due to rock pressure, and are useful in foretelling rock bursts, and whether it is safe to remove supporting pillars. Rocks containing sericite or talc in their joints, which act as a lubricant, do not make as much noise as other rocks.

Accidents while blasting may be due to defective fuses, or when the fuse has been cut too short to allow the miner to get away, or a miner may return to the face too soon after a hang-fire. When about to fire, and the men have retired, a man should be stationed near any entrance to where the blasting is about to take place, to warn anyone approaching.

Men sometimes get pinched between a moving truck and timber sets. This should never occur with hand trucking, but does sometimes happen with mechanical haulage which is quicker. Refuge holes are made at suitable distances apart, and men should be able to judge the distance away of trucks by the sound they make.

Falls down shafts and winzes are mostly due to horse-play or carelessness. Adequate guards should be provided for such places.

In a two-compartment pass, one for ore, the other for a travelling way, the man shovelling ore into the ore compartment may accidentally or on purpose shovel some into the travelling compartment when someone is climbing the ladders.

This might be prevented by having a suitable cover over the travelling way, and the man about to travel should call out to those in the stope that he is about to climb up. A trucker by standing in the wrong position when freeing a stuck up pass, may be hit by the bar he is using, or by the ore, if it comes out with a rush.

Only safe methods of mining should be employed. For instance if there is a large wedge-shaped block between levels to be extracted, it is specially dangerous, since it is free above and below and its only support is the sides. Such a block should be attacked from the toe. The miner then has a good roof overhead. If attacked from the other end it is only a matter of time before the walls are unable to support the weight, and it will fall on those working below.

There should always be at least two exits from a mine by shafts, adits, or through old workings, for if there were only one, and that became blocked, those below would become entombed.

When there is an explosion in a coal mine there are generally more deaths due to poisonous gases than to the violence of the explosion.

Accidents to Property

Accidents to property too often include accidents to the workers: they also generally throw a certain number of men out of employment for a time.

Though coal and metal mines have certain conditions in common, they differ in other respects so that methods of treatment are not necessarily the same. Metal mines, as a rule, are not so extensive as coal mines, they are more inclined to be vertical and are generally broken through to the surface along the outcrop of the deposit.

Fires. Fires may occur in either metal or coal mines, and may be brought about by spontaneous combustion; using naked lights near inflammable material; placing a lighted pipe on timber; leaving a lighted candle snuff on timber the wick of which may fall out; tipping tobacco ashes among combustible rubbish; making a fire to warm tea against timber; by incendiarism; or short circuiting of electric wires. Timber in the neighbourhood of steam pipes is predisposed to ignition.

To prevent mine fires, certain precautions should be taken. The entrance to mines should be concreted to prevent surface fires from spreading to underground workings. Long flame explosives should not be allowed in collieries. Air pipes for drills should be so arranged that in case of fire water can be diverted into them. Fire hoses on portable reels with nozzles attached might be kept in recesses along entries at intervals of 600 to 1000 ft. For underground purposes a 1½ in. hose has been found to be the best size with a ½ to ⅜ in. cone nozzle. Have valves placed on the rising main of the pumping plant at each plat so that mine water can be used in case of a shaft fire. Shaft fires are especially dangerous, because they affect the means of escape. The pressure of water should be reduced in deep mines, where the head is too great, by reducing valves or some other means to less than 75 lb/sq in. at the hose, so that one man can control it. Pump chambers, motor rooms, and stables should have an incombustible lining: rubbish should not be allowed to accumulate, and oily waste should be placed in covered metal containers. Oil and other combustible material should be stored in a specially isolated place. Acetylene lamps are safer than candles or oil lamps as they have no glowing wicks, and are extinguished of their own accord.

When a fire breaks out, a general mine warning should be given to those below. This may be done by adding some odoriferous non-toxic chemical to the compressed air pipe such as ethyl-mercaptan which has an odour of rotting cabbages; switching electric lights on and off, or some other prearranged signal. In a colliery the odour of gob stink, like that given off when a coal fire is first lit, gives warning. Every fire has a small beginning. If caught in its early stages it may often be filled out, covered with earth, or extinguished with water.

Selected men should be trained for fire fighting and rescue work, and broken in to the use of breathing apparatus containing compressed or liquid oxygen, and an absorbent for carbon dioxide. All men are not suitable for this kind of work. In a hazy atmosphere, the rescuers should hold a light manilla guide rope, and use electric lamps.

In metal mines superheated steam and inert gases, such as carbon dioxide and sulphur dioxide, have been used but without much success. The steam soon condenses, while the inert gases are not generated in sufficient quantities to fill a large area. Carbon tetrachloride is useful for reaching a fire in a confined space, also for electric fires, as unlike water it is a non-conductor of electricity; but it should not be used in quantities more than a quart in one place underground for fear of men being overcome by the gas.

Timber burns more fiercely than coal, but does not give off explosive gases like coal; however they give off pyroligneous gas which has the same effect on man as laughing gas, according to his nature, and causes him to laugh, weep, or to become pugnacious. In square set stopes a water screen may be set up to cut off the flames by having troughs in the bottom of which holes are bored; water is then led into the troughs which rain down.

Spontaneous combustion is due to chemical action and occurs more frequently with fine material as the latter offers a larger surface than that which is coarser. The chemical action is assisted by moisture. In metal mines where marcasite exists, any broken material should be filled out at once, but if a fire is burning, it may be extinguished by gently blowing the products of combustion back on the fire, so suffocating it. In coal mines, the general opinion is that spontaneous combustion is due to the occlusion of oxygen by the coal. In colliery fires gases are distilled off which burn if supplied with sufficient air.

If a colliery fire gets a good hold, it must be sealed off as close as possible to the seat of the fire. At first a temporary stopping that can be quickly erected, such as brattice cloth; rails with sheets of iron placed against them; wooden planks, the joints caulked with clay help to prevent air from feeding the fire. When there is no fear of further explosions, a permanent stopping is built of sacks filled with sand or damp clay, two or three thick; bricks, or concrete are also used. These should be let into recesses in the walls and roof, and a pipe with an automatic valve is inserted to allow gases under pressure to escape into the return airway. Samples of the air are taken periodically, and also the temperature and pressure. Stoppings are started on the intake side first, where the men can work in fresh air. Stopping of the return side is more dangerous in coal mines, where explosive gases may be backed up and result in explosions, than on the intake side. In spite of all precautions, there will be a leakage of air through the strata. Mine fans should be made reversible, but should never be reversed except by order of the mine manager for, if all the lights have been extinguished by an explosion, the men depend on the direction of the air current which they are accustomed to for guidance to safety.

As a last resort, a mine may be flooded, if below water level, and there is sufficient water available. Some parts of a mine are higher than others, and air will collect in such places compressed by the head of water: this air will help to keep the fire alight though it may be extinguished in other places. If correct plans of the workings have been kept, bore holes should be put down so as to tap the air in high places and set free the imprisoned air. A fire may be extinguished for want of air, but the coal may be so hot that as soon as the workings are opened up again, the air admitted will restart the fire. Water, besides extinguishing the fire, softens the rock so that it falls in, loosening the timber, and this causes much time and expense in carrying out repairs. If water is not available, a cutting may be made all round the fire and the space solidly filled with clay to prevent the fire spreading. Collieries working on the panel system can continue working when the panel in which the fire is burning has been sealed off. The surest way to extinguish an underground fire, where conditions permit, and other means fail, is to fill the workings with slurry containing 40 per cent water which is sent below through 2 in. boreholes or pipes.

Watering to lay coal dust in collieries is no good unless the dust is thoroughly saturated. The finest and most dangerous dust settles on the timber overhead. To prevent the flame caused by an explosion from spreading, rock dusting is employed. The rock used is generally shale or limestone which is powdered so that 100 per cent will pass through 20 mesh to the linear inch, and of this 50 per cent should pass through a 200 mesh. The amount of dust used should be not less than 100 lb/sq ft of the cross-section of the entry. This dust is distributed in zones by hand or hand worked blower. Dusting must be renewed every few days as fresh coal dust is constantly being formed. If an explosion takes place, the force stirs up the dust which acts as a screen and prevents the flame passing through it. In metal mines, fire doors may be placed at plats which can be closed from the surface in case of fire when it is known all the men are out, and so cut off a fire from the shaft or workings.

Buildings about a mine should be isolated to prevent a fire from spreading. Hydrants should be placed near all main buildings, also near the entrance to a mine. If water is not available, chemical extinguishers should be used. All inflammable material should be kept away from shafts. Oil and explosives should be stored in separate buildings well away from other structures. Surface fires about a mine are dealt with in the usual way.

Underground fires may be caused by short-circuiting of electric wires; leaving a lighted candle snuff or tobacco pipe on timber; shaking tobacco ashes or throwing a cigarette butt on to inflammable material; lighting a fire to boil water too near timber; blasting too near to timber, etc. The necessity of preventing such occurrences is obvious.

Fires may also be caused by spontaneous combustion, as when mining certain classes of coal or marcasite.

The existence of a fire and its nature may be indicated by the odour given off, as that of gob stink in a coal mine; of sulphur dioxide when sulphides are burning; and the smell of burning wood when sets of timber catch alight.

Once a fire has started, there is no time to lose. At first, when limited in extent, the material may be brought out in iron trucks if approachable. If this is not possible, water may be played on it if available, or some chemical extinguisher used. Superheated steam has been used successfully for extinguishing small fires before the steam could condense. Carbon dioxide generated at the surface has been used to suffocate fire, but the gas is very penetrating, and is likely to drive men from parts of the mine not affected by the fire. In some cases the products of combustion may be blown back gently on to the fire, care being taken that the fire is not supplied with fresh air. A 10-in. blower, mounted on a truck and worked by electricity to be mobile, is useful in forcing back products of combustion. If a fire has a good hold, it may be necessary to seal off the affected portion of the mine to prevent the fire from spreading. Sulphur dioxide cannot form till the temperature of the mineral is above 100° C., when all the heat available converts any water present into steam.

The treatment of a mine fire will depend on the nature of the fire, its extent, its position, and the means available for attacking it. Timber fires burn and spread more rapidly than coal fires. Mines dependent on natural ventilation are not so safe as those ventilated by mechanical means, because the fire may cause a reversal of the air current.

All underground officials should know the position of valves, electric switches, and what action to take in case of fire; first to save the lives of the men, and secondly to save property. When the fresh air supply is restricted, an underground fire tends to travel against the air current in search of oxygen; but if the current is very strong, the fire may travel with it. All electric equipment should be placed in fire-proof stations.

Certain precautions should be taken to prevent fires, or if started to nip them in the bud. Oily waste should be placed in metal containers and covered. Rubbish should not be allowed to accumulate, especially near the entrance to a mine. All blasting operations should be properly carried out. In certain cases where mine timber is liable to ignition it should be made fire-proof with gunite. Under no circumstances should a miner be allowed to break down more coal than he can clean up and load out before leaving his shift. The same applies to the extraction of marcasite. No naked light, matches, pipes, or tobacco must be allowed in a gaseous mine. Greasing of trucks under ground should be done in a place where, should a fire occur, it can do no harm. Neither smoking nor open lights should be allowed in a cage. Shaft men should use electric lamps. Buckets of water and sand should be kept full at suitable places where they are likely to be required to subdue small fires, the sand being used to absorb and smother oil fires. Foreigners who do not understand English should not be employed in or about mines in English-speaking countries.

There are three forms of chemical extinguishers—

1. The ordinary solution of bicarbonate of soda with a glass vessel of sulphuric acid of about 58° Baumé, which can be broken by a thumbscrew when required to be brought into action.

2. Carbon tetrachloride, which at a temperature of about 200° F. is transformed into a heavy gas which smothers a fire. It is especially useful for electric fires; but it is not advisable to use a large quantity underground in a poorly ventilated place, say more than a quart, for fear of men being overcome by the gases generated.

3. Foamite, which generally consists of bicarbonate of soda, sulphate of aluminium, and a small proportion of extract of liquorice, is specially good for oil fires.

There should be a fire door on each plat of a metal mine and near the top of a shaft. The object of a fire door is not to stop a fire, but to check the current of air. An iron door would buckle with heat. A fire door at every plat will serve to prevent a fire in the workings from reaching the shaft. Such doors can be made to close from the surface by pneumatic means. A fire door near the collar of a shaft is useful in case of a surface fire. Usually the posts of the second set are taken out and a layer of concrete placed on the top of the second shaft set. On this, concrete pillars are built to take the place of the wooden posts removed. Shaft doors of ⅜ in. iron plate are carried on hinges set in the concrete over the wall plates and closed to meet at the centre line of the shaft, being shaped to pass tightly round the guides when closed. These doors can be closed from the surface by means of ropes. A perforated 3½ in. pipe is placed a set below the fire door to act as a sprinkler.

Shaft fires are especially dangerous as affecting the way of escape. If the fire is in a downcast shaft, the gases will soon be distributed throughout the workings. The draught is not likely to be so great in a stope or winze where the current is split up. A timbered shaft, even if damp, will burn readily when once a fire gets a good hold.

Water-spray arrangements are sometimes made in shafts. A pipe-line is carried down the shaft and connected with sprays set under the timbers about 25 ft apart. A uniform flow is obtained through all the sprays by adjusting service cocks in the line connecting each spray with the main. The water is turned on only from the surface. Fusible plugs are not advisable for shaft sprinklers, for if started at the wrong time they might endanger the lives of men underground by reversing the current of air; but fusible plugs at plats are all right.

To attack a fire in a stope, commence at one end and gradually seal up all approaches except one. Then start from the other end, and if necessary use a fan to drive the products of combustion back on the fire, and seal up all approaches. When this is done, return to the first end and seal up the last exit. If the stope is filled with square-set timber, pyroligneous fumes are distilled off which has a similar effect on men as laughing gas. Sometimes a water curtain is formed to keep the fire from spreading in stopes. This consists of a trough with a perforated bottom into which water is led. To pour water into a mine is not much good unless it is desired to flood it, as usually the water just flows down the footwall and does not reach the fire. Surface cracks caused by the collapse of workings will admit air to an underground fire. Such cracks should be filled up with slime.

The most effective way to extinguish an underground fire is by hydraulic filling. Sand and clay are mixed in a tank with about 40 per cent of water by a horizontal perforated pipe at the bottom, through which air is forced. The product is sent to the seat of the fire through pipes or diamond drill holes. Clay fills up small cavities better than sand, but water does not drain off from it so well as from sand. Excess water is pumped to the surface. If this water is hot, it shows it has reached the seat of the fire. If later on the water is cool, it shows its object has been attained.

When a fire breaks out in a coal mine, it may be necessary to isolate it by building barricades. Some recommend the closing of openings to the return passages first; but this is difficult on account of the gases and smoke given off.

The general opinion is in favour of closing the openings on the intake side first, to cut off the supply of air to the fire. The place where a barricade is being built must be well ventilated and the men should work in relays to prevent over exposure to fumes. In some cases breathing apparatus has to be used. War masks are of no use, as mine gases are of a different nature to those used in warfare. A temporary bulkhead can be built by resting sheets of galvanized iron against bars placed across the passage and filling the space between the iron and rock with moist clay. This gives the miners an opportunity to erect a stronger and tighter one a little distance away which may consist of 2 in. boards nailed to posts, cap piece or collar, and fitted as closely as possible against the sides. Two or more strips 3 × 3 in. or 4 × 4 in. are nailed to the outside of these, and another board wall is nailed to the strips, moist clay being tamped between the walls as the outer one is being built up. The outside wall should be carefully caulked, especially round the edges.

Permanent barriers may be constructed of brick or concrete, varying from 20 in. to 5 ft in thickness, according to circumstances. The outer surface is plastered with cement from time to time.

To sample air in the fire area periodically, a 3 in. iron pipe is led through the barricade near the top, and a 5 ft glass tube is passed through this so as to go past the influence of the outside air which might be drawn in. The rest of the space round the glass tube is closed with a rubber bung. Air from inside the bulkhead is aspirated into a Winchester quart bottle which is then removed to the surface to determine the amount of carbon monoxide, carbon dioxide, methane, and oxygen present. The temperature is also taken by means of a thermometer. Automatic valves of ample capacity should be provided to pass the gases from the fire to the return airway when the pressure rises above the fixed maximum as shown by a gauge. When the seat of the fire becomes surrounded with an atmosphere containing as little as 12 to 15 per cent oxygen, flames subside, but combustion continues until the oxygen is reduced to 5 or 6 per cent.

All fires are deceptive and are likely to break out again when thought to be extinguished, till air is brought in contact with the hot material, as there is no chance for the heat to escape.

Where water is laid on in a mine, suitable connections for fire hoses should be provided at stated intervals so as to reduce the length of hose to be handled. In deep mines where the head of water is excessive, pressure-reducing valves or some other means such as open tanks at certain levels should be introduced to hold the pressure less than 75 lb per sq in. at the hose, in order that one man may be able to control it.

If the decision is taken to flood a coal-mine as a last resource, a rough idea of the quantity of water required may be obtained from the knowledge of how much coal has been extracted, and the extent of the workings, allowing a cubic yard of water per ton of coal. An objection to flooding a mine, if it can possibly be avoided, is that water softens the rock causing the timber supports to fall in and the unsupported roof to cave; so when reopening the mine most of the travelling ways are in broken ground.

When reopening that portion of a mine which has been alight and isolated, see that the pressure on each side of the barrier is the same before breaking through, to prevent the men being asphyxiated by an outburst of fumes. The barricades on the return side should be opened before those of the intake.

The signal for "All out" may be made by switching off and on of electricity, where electricity is used. Stink signals are used in places where compressed air is employed. The stink substance may be ethyl mercaptan, which smells like rotten cabbage, but is not poisonous like sulphuretted hydrogen; or eucalyptus, which is let into the compressed-air system. The regular ventilation system is insufficient to carry warning stenches.

Large heaps of coal over 12 ft high should not be stacked at the surface. Should such a heap catch alight, get a 2 or 2½ in. wrought iron pipe of suitable length, close one end to form a point. Drill ⅜ in. holes in a spiral form for its entire length. The point is then driven into the heap to the seat of the fire. A hose is attached to the open end and water forced through it. Several such pipes may be necessary.

In large mines there should be a team of five reliable, thoroughly-trained men for each shift, so that a team of experienced men is always available. The men should be carefully selected, as all men are not suited for fire fighting, temperamentally or physically. Comparatively young, slight wiry men are generally found better than more heavily-built men. They should live near the mine, so that they can be called on at short notice. There should be frequent practices, using fire equipment, such as the Proto, as there is no time to teach when a fire is on. The team should be familiar with the mine workings. There should be two or three extra trained men available in case any of the regular team are not available. If there is a group of small mines which cannot afford to maintain individual fire teams, they may combine to support one available for any of the group when required.

Flooding. Water may find its way into a mine from the surface or from some underground source. To keep out surface water, the entrance to a mine, whether by adit or shaft, should, when possible, be above flood level, and the underground workings should not be connected with the surface.

With regard to an influx of water from some underground source, this may be due to accidental tapping of old workings full of water. If unsurveyed old workings are known to be in the neighbourhood of present-day workings, bore holes should be drilled in all directions, and wooden plugs kept at hand to block a hole should water be tapped. In deep alluvial deposits which are full of water, bore holes are put up from a rock level underneath it, so as to drain the water in quantities suitable for the capacity of the pumps.

Coal seams and other beds which pass under swamps, lakes, or seas may strike a depression in the cover and flood the workings. There should be a strong pillar between the underwater workings and those under dry land. Any openings through the barrier should be as small as conditions permit, and capable of being closed quickly with watertight doors. There should also be a barrier pillar between a mine working under a large body of water and its neighbour, so that in case of an accidental break through the neighbour will not be affected.

Mine flooding may be due to—

1. Surface waters.
2. Water bursting in from old workings.
3. Mining under large bodies of water—swamp, lake, river, or sea.
4. Mining under deep alluvial deposits which have to be drained.

(*1*) All entrances to a mine should be well above the highest flood mark. It sometimes happens that this is not possible, and a shaft has to be built-up above the natural level of the ground. Most metal mines have several openings along their outcrop which cannot well be made water-tight. In those mines where the mineral deposit does not come to the surface in low-lying ground and the only entrance is through a shaft, arrangements should be made to disconnect pipes and wires near the mouth of the shaft, and to have concrete covers kept in recesses near the surface, under ordinary conditions, which can be thrown across the shaft and held there in time of flood. In spite of all precautions, there will be a certain amount of water leakage which will compress air in the workings. An escape for this air must be provided, through a pipe of sufficient size, let into the shaft below the seal and carried up the headframe; otherwise pressure of air may displace the seal and damage the headframe.

(*2*) Mine flooding may be due to breaking into old workings containing water. If there are no reliable surveys of the old workings available, advanced bore holes must be drilled in all directions for at least 5 yd, and wooden plugs be handy to stop the flow of water if struck unexpectedly. An indication of approaching old workings may be given by water seeping through joints in the rock if there is any head of water. When it is known that water will be struck in a certain hole, a pipe with a gate valve is inserted so that the flow can be regulated to suit the capacity of the pump.

(*3*) When mining under a large body of water, the ground should be worked in sections or panels, separated by strong barrier pillars with the least possible number of openings into each panel which must be supplied with strong watertight doors that can be readily closed. A roof of solid rock must be left of suitable thickness, not less than 120 ft, depending on the weight of water, loose sand and silt overhead. In addition to this, allowance must be made for the composition of the overhead rock, its strength, dip, its liability to disintegrate on exposure to the air, depressions on the bed of the water, the presence of faults and other signs of weakness, the thickness of the mineral to be extracted and the ability of the pillars left for support to resist rupture or squeeze. The thickness of rock overhead can be determined by comparing bores through sand and silt to solid rock with underground surveys. To compensate for errors in vertical surveys, an allowance of not less than 5 ft in thickness of rock should be made. To compensate for possible depressions, the allowance may be 35 ft.

(*4*) In deep alluvial mining, a main drift is driven several feet below the alluvial which is generally saturated with water that has to be drained off before the wash can be mined. To do this, lateral drifts are driven on either side of the main drift and bore holes are jumped up from these through a piece of pipe provided with a slide valve, which can be quickly closed or regulated according to the capacity of the pumps. All drifts have a very slight grade towards the shaft so as to give the miners a better chance to escape before the entrance to the shaft is closed by water. Water-tight doors are provided at suitable places to dam back excessive water. The frames of these doors are provided with drain holes which can be closed by valves, and there is a pipe near the roof to allow air compressed by water to escape which is closed by a valve when water reaches its exit. If slurry is expected, a strong wooden door is made in two horizontal sections. When the lower section is closed to hold back the slurry from reaching the shaft, it gives any trapped men a chance to wade through it and climb over the top before the upper section is closed.

Bursting Ground. There are earth waves as well as waves of water and air. The effect of these can be seen in the elevation and submergence of land at the present day, and in past times. Earth strains may be relieved over a large area in the form of joints, but if the rocks are strong they yield but little before breaking and then suddenly causing a displacement known as a fault. Rock bursts may be due to horizontal or vertical pressure, and may occur near the surface or more frequently at a depth. They generally occur in stopes but may occur in other openings some distance from stopes. One cannot prevent rock strains, but they can be controlled to a certain extent. If a burst is expected after blasting, the place should be given a period of rest to allow things to settle.

When an excavation is made, equilibrium is destroyed and the compressed rock will expand into the excavation. One should avoid the merging of two large excavations at a depth. In some places under strain which has not yet been relieved, and if an artificial cavity is made, there is likely to be a rock burst, especially when approaching a higher level and the arching action of the bridge does not give sufficient support. In such a case, the open excavation should be small, so as to give the walls as much support as possible. Parallel veins should be stoped singly; that in the hanging wall being first removed. Where veins branch off, commence extraction at their intersection and then work one vein at a time.

Change House

If in a cold climate, the change house should be connected with the travelling shaft by a fireproof-passage so that men coming out of a hot mine shall not get a chill. The house should be spacious, especially if there is likely to be an increase in the number of miners. It must be well ventilated.

but free from draughts. The windows should be sufficiently high to prevent outsiders from looking in. The floor should be concreted and given a suitable grade for draining. A special man must be in charge to keep the place clean and see that no unauthorized person enters.

The bathroom should be provided with wooden mats. Hot and cold water should be laid on, the hot water being heated in a boiler outside the building. One basin and shower should be provided for every eight men in a shift, for they will not all be wanted at the same time. The showers and washbasins or troughs should be in a different room from that in which the men change their clothes. The basins should have press taps to prevent waste of water. The temperature of the room should be kept at 75° F. Each man has two lockers, one for street clothes, the other for working clothes. The latter is open at the top and has a hotwater or steam pipe underneath for drying them. The lockers for street clothes are best arranged on one side of a passage, and those for working clothes opposite. Seats are provided in front of the lockers. Or instead of lockers, each man may have a metal bucket in which to place his boots, and hooks attached on which to hang his clothes. These are hoisted up out of reach from the ground by a rope which can be secured by a padlock. Mine clothes must be taken home every week to be washed. At one end of the change house are three rooms. 1. A change room for the officials. 2. An office for underground bosses. 3. An office for the timekeeper, also a place where candles or lamps can be given out.

Water

In places where the only water is salty or otherwise impure, it should be distilled for drinking purposes. This may be done on a fairly large scale by heating the impure water in iron tanks or boilers and leading the steam to a series of large inclined galvanized iron pipes, where the steam is condensed by the air and collected in containers.

In hot mines where men sweat a great deal and drink much water to make up the loss, they are liable to get weak owing to the loss of salt from their blood. To make up for this loss their drinking water should have a little salt placed in it.

Septic Tanks

Septic tanks are used to purify sewage by means of bacteria. They are suited for places where there is no proper drainage, if there is sufficient fresh water. The method requires two tanks, preferably built of concrete, of sufficient capacity to treat the sewage for 24 hours. All pipes in its structure should be of glazed earthenware, since metal pipes, such as copper, zinc, iron or lead, unless kept coated with tar, quickly get eaten through. The sewage first passes into a tank which has a cover and an air vent, and there is an inspection pipe connected with the pipe from the closet. This tank is connected with the second tank by a siphon, and there is also a discharge pipe. The inflow, siphon, and discharge are all on the same water level. In the first tank anaerobic bacteria which work best in total darkness and without air, act on the sewage. They form a scum on the top of the liquid which reaches a certain thickness, and sludge in the bottom. It is here that the bacteria attack the solid matter and dissolve it. When liquefied, the fluid passes through the siphon into the second tank where it is further decomposed by aerobic bacteria which work best with plenty of both air and light. The final liquid which

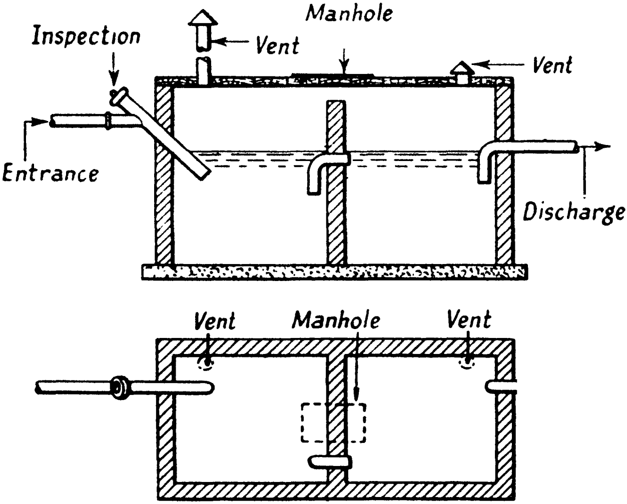


FIG. 67. SEPTIC TANK

escapes is quite harmless and may be allowed to soak into the soil. Paper, straw, and such rubbish, if much is present, should be filtered off beforehand.

An excess of water such as drain water from roofs should not be allowed to enter the tanks, neither should greasy water from kitchens pass into septic tanks. At first there may be a disagreeable smell from the tanks, till the scum is formed. Disinfectants, chemicals, etc., which might destroy the bacteria must not be allowed to enter the tank, and for this reason mine water is generally harmful to the good working of the tanks. The inlet and outlet should be so arranged as to allow the maximum of distribution with a minimum of movement of the fluid so as not to interfere with the surface scum.

Closets

Underground, and also on the surface, where there is no proper system of drainage, or sufficient water for septic tanks, night pans are best used. These pans are of galvanized or enamelled iron, 12 to 13 in. high, provided with handles and well-fitting covers. They should fit as closely as possible under the seat which is 14 in. off the ground, and the hole should be oval in shape with the edges moderately rounded. The pans are generally changed from the back of the seat where they are guided to the proper place by runners. Earth should be supplied to cover the excreta and keep flies away. Disinfectants should not be used in the pans, as that would prevent the excreta from decomposing. The night soil should be buried in trenches about 10 in. deep, where it is soon decomposed. The trenches should be dug systematically about a foot apart. Sometimes closets are not divided up; the idea being that the men will not remain longer than absolutely necessary.

MINE ORGANIZATION

A mine should be neither over- nor under-staffed. In small mines the same official may have to undertake different duties; the surveyor may also act as sampler; or the accountant as storekeeper, but as work increases more officials will have to be employed.

The staff of a mine should be so organized that different branches do not clash: they should work as a team, and while each branch should do its best, it must not be done at the

SCHEME FOR THE DIVISION OF WORK

GENERAL MANAGER

OFFICE

Chief Accountant—

1. ***Head Office*:**
   - Assistant Accountant
   - Chief Book-keeper
   - Cost Clerk
   - Invoice Clerk
   - General Clerks
   - Messengers
2. ***Correspondence Dept.*:**
   - Stenographer
   - Filing Clerk
3. ***Time Office*:**
   - Paymaster
   - Timekeeper
   - Wages Clerk
4. ***Store*:**
   - Store Buyer
   - Storekeeper
   - Assistant
5. ***Traffic Dept.*:**
   - Traffic Clerk
   - Weighman

MINE

Mine Manager:

1. Surveyor
   - Assistant
   - Sampler
2. Underground Foreman
   - Shift Bosses
   - Timber Boss
   - Trucker Boss
   - Mullock Boss
   - Pumpman
   - Pipeman
   - Tracklayer
   - Bracemen
   - Sawyer

TREATMENT PLANT

Metallurgist:

1. Chief Assayer
   - Assistants
2. Sample Mill Foreman
3. Mill Foreman
4. Smelter Foreman
   - Converter Foreman
5. Yard Foreman

ENGINEERING

Chief Engineer—

1. Mechanical Engineer
   - Machine Shop
   - Pattern Shop
   - Foundry
   - Blacksmith
   - Boiler-maker
   - Sailor-gang
2. Electrical Engineer
3. Designing Engineer
   - Draughtsmen
4. Constructional Engineer
5. Carpenter Foreman
   - Builders
6. Surface Foreman
   - Labourers
   - Bricklayers
7. Fire Department

expense of another. Each officer should be made responsible for the work under his charge. Each under-officer should be so trained that he could fill the place of his superior in case of necessity (should he be sick, away on holiday, or leave). Officers should be carefully selected, not only for their knowledge but also for their temperament. If a man is temporarily transferred from one department to another, he is solely under the control of the department to which he has been transferred, except when he requires technical help.

The scheme shown in page 332 is only suggestive and should be modified according to local requirements.

Proportional Work

The following notes based on an eight-hour day should be taken proportionally for the same class of man, since the work a man can do varies with his physique, health, nationality, etc.

To pick up and throw down close by, one-man stones, a labourer should average about 27 cu yd loose stone: or, say, 16 cu yd in the solid.

A man will load or unload one-man stones into or out of a high-sided wagon at the rate of 13½ cu yd loose: equal to 8 cu yd solid.

A man will load into a wheelbarrow stone already sledged, at the rate of 17 cu yd loose; equal to 10 cu yd solid.

Shovelling ore crushed to pass through a 2 in. ring when in a heap without a smooth bottom, one good shoveller can shift 11 cu yd loose; equal to 6½ cu yd solid.

When there is a flat bottom to shovel off, a man can shift 16 cu yd loose; equal to 9½ cu yd solid.

Small stones ¾ in. are readily penetrated by a shovel, so do not require shovelling boards.

A wheelbarrow load averages $\frac{1}{25}$ cu yd solid rock and takes 2 min to load. A man will wheel it at the rate of 180 to 250 ft/min, depending on whether he is lazy or active, and he occupies ¾ min each trip in dumping the load, fixing run planks, etc.

A team of horses pulling a wagon over an ordinary earthen road will travel 2½ m.p.h.: if over soft roads only 2 m.p.h. Over the best macadamized roads they will travel 3 m.p.h. When the hauls are short, the actual hours of walking time may be considerably lessened on account of the time lost in loading. Snatch teams should be used over bad hilly or boggy spots.

Classification of Expenses into those due to Revenue and those due to Capital and Suspense

H. C. Hoover tabulates them as follows—

Revenue	(1) General expenses		
	(2) Ore extraction	Ore breaking Supporting stopes Trucking ore Hoisting	Various expenditure for labour, supplies, power repairs, etc., worked out per ton, or foot advanced over each department
	(3) Pumping and ventilation		
Capital or Suspense	(4) Development	Shaft sinking Station cutting Cross cutting Driving Rising Winzing Diamond drilling	
	(5) Construction and equipment	Various works	

Economics

A mine is a wasting asset; it gets worked out and the ore is not replaced. A mill gets worn out. Both mine and mill may be capable of expansion: the mine by development: the mill by additional units as the mine is developed. The greater speed with which a mine can be developed to its maximum output is the most economical. The object of mining is to get the greatest profit from a given body of ore. The sooner one can get returns on the capital spent the better. The maximum output is necessary for the cheapest production, since expenses in management and administration are going on all the time. When considering the advisability of extending the original plant, the following factors should be taken into account—

1. The cost of production as affected by the increased output.
2. The amortization or redemption of capital invested in the additional equipment, with accumulated interest thereon.
3. Limitation imposed by the uncertainty of continuity in depth.

Working expenses may be divided into those which vary with the tonnage, e.g. development, haulage, and treatment, and those known as "fixed," which depend partly on time as well as on tonnage, e.g. pumping and management.

By working on a large scale there are certain economies in working expenses; better terms may be obtained for freight, larger quantities of stores can be purchased at a reduced price. The profits will also be enhanced by the interest earned on the extra profit accruing on account of the time factor. All these additions of profit as the result of expanded output (not including the ordinary profit on ore) may be considered as "increment of profits." As the life of a mine and plant dies, the capital expended on it must be paid back with compound interest during the existence of the mine.

"If by vigorous development, the visible life of a mine as shown by the ore reserves is lengthened so as to exceed the time required for a unit of treatment capacity to earn an amount through the increment of profits equal to the amortization of the capital invested in that unit, then the installation of another unit becomes not only justifiable but an obligation of good management.

"The economic and advisable ore reserves should be equal in volume to the annual output multiplied by the number of years just under that needed by the increment of profits to equalize the amortization required to construct increased treatment units.

"An increase in the mill is justified when the redemption of the capital involved, plus interest on it, can be more than balanced by the saving in cost or the increment of profit upon the additional tonnage taken from new ore reserves. . . ." (H. C. Hoover.)

Unfortunately, some managers do not go into figures and increase their surface plant before they are justified by the development of the mine.

Development should keep pace with the depletion of the deposit. When the price of the metal being mined is unusually high, it may be well to put on all hands extracting ore and to suspend development for the time being. Or climatic conditions may make this desirable, but as a rule it is better to extract and develop regularly, especially if a good team of men has been got together.

Every ton of ore extracted should be debited with its proportion of development.

Ore extracted during development should not be looked upon as having cost nothing and be added to the ore stoped, making the cost of the latter look so much less. It should be charged for at the cost of stoping and credited to the cost of development.

Mining Accounts and Costs

Every year or half-year a Company should publish a report which includes a Balance Sheet, Profit and Loss Account, and sometimes a Working Account, together with other data that should permit a shareholder or intending shareholder to size up the position of the Company.

Frequently the information supplied is too meagre to enable one to analyse the figures as desired, so that further details may have to be sought, which may or may not be available. Certain headings may convey no meaning to a critic. Some accounts are purposely lumped together with the object of masking individual profits or losses. As a matter of fact the ordinary Balance Sheet of a Mining Company is scarcely worth the paper it is printed on, and in order to obtain even an approximate idea of the value of the property other information is necessary, some of which is obtained in the Report, while some must be sought elsewhere. The comparison with previous Reports often supplies required data, and certainly assists one in becoming better acquainted with the condition of things. Sometimes the excuse is given by Directors that it is not for the benefit of the Company to make certain information public for fear competitors should make use of it. It is the part of a competitor's business to know what others in his line are doing, and knowing his own costs, it is not a very difficult matter to ascertain, near enough, the costs of others; while the shareholder, not having this knowledge, is kept in the dark. A shareholder wishes to ascertain if his capital is safe and what chance there is of legitimately earned profits being paid at stated times.

In order to get a fair insight into the real financial position of a Mining Company it is necessary to consider the items in detail and classify them in a proper way. One must ascertain what assets have been hypothecated to secure special creditors; what is available for the benefit of unsecured creditors; and after discharging all liabilities, what is left. One should ascertain whether the interest accrued to the date of the Balance Sheet on secured loans is included amongst the liabilities under the heading relating to secured creditors. Then one wants to know the amount due to creditors for goods supplied in the ordinary course of business. The more detail that is given in an annual report the better position one is in to determine the true position of affairs, but as this is likely to lead to questions, essential points are frequently omitted. Besides the total expenses and receipts, and the profit or loss that has been made, one wants to know how the profit or loss has been earned or made, whether it is likely to continue, and if so for how long, what changes may be expected, probable variations in metal market, prospect of labour troubles or lawsuits, life of mine, whether returns are given in long or short tons, for calendar or lunar months, whether the season has been exceptionally favourable or adverse, etc. Figures extending over a long period are more reliable than those for a short time.

The Balance Sheet is a collection of balances left on the Trial Balance after the Profit and Loss Account items have been extracted, and should state the financial position of the Company, i.e. what it is worth. Often the Credit side of a Balance Sheet is headed "Assets" and the debit side "Liabilities." As the term "Asset" is now used in business to mean all available property and rights which can be applied in satisfaction of the liabilities, or which can be turned into money or money's worth, it is better not to use this word here, since a number of items on the Credit side of Balance Sheets are neither valuable property nor rights, and cannot be applied in satisfaction of liabilities or turned into money or money's worth.

A mine is a wasting asset, yet it is usual to see the original purchase price of the property passed on from year to year, with no attempt to write off the proper proportion, according to the assumed life of the mine. Besides the Mines Property Account, on the Credit side of the Balance Sheet, we have such headings as Mine Development, Mine Plant and Buildings, Office Furniture and Fittings, Stores on Hand, Reserve Fund Investment, Dividend Suspense Account, Ore Concentrates and Matte on Hand or in Transit (at estimated net value, less advance payments), Sundry Debtors, Cash in Hand, Balance at Bank, and the Deficiency on Profit and Loss Account, if any. The Mine Development Account is the money which has been spent on developing the mine before it reached the producing stage, but should not include among preliminary expenses such items as commissions paid on issue of shares or debentures or discount given in respect of the issue of debentures, these being shown separately, and this has to be written off by degrees as Development Redemption.

When once the mine becomes productive, the cost of development may either be added to the Mine Development

Account and a certain proportion charged to each ton mined, or the mining has to stand the cost of all the fresh development done during the given period. The Plant and Building, and Furniture and Fittings Accounts should have depreciation written off every period. The depreciation should be so apportioned that the cost is nearly wiped off at the termination of the life of the mine, for the market value of the asset is very small by that time. If the life of a mine is long, some of the plant is out of date and has to be scrapped before it is worn out, for if taken care of, the machinery should in most cases be workable after many years of use. If the life of a mine is taken at, say, ten years, and 10 per cent is written off for depreciation every year, the amount will not be redeemed in ten years, even if no additions are made in the meanwhile, for it is usual to write the 10 per cent off the asset remaining, not 10 per cent of the original asset. When comparing Balance Sheets of different years, one should note if preliminary and other expenses are being gradually written off; if not, it is probably because the profit is too small to do this and pay dividends, or because there are no profits against which to charge it.

Before the producing stage of a mine is reached, all expenditure should be charged to Capital Account, e.g. (1) Property, (2) Main Shafts and Adits, (3) Underground Development (including shafts sunk on ore), (4) Machinery and Plant, (5) Buildings, (6) Surface Work (reservoirs, railway sidings, roads, etc.). (7) General expenditure (head office charges, etc.) Any Sundry Revenue received during this stage should be deducted from General Expenditure, and the balance distributed proportionally over the remaining heads of expenditure. Main Shafts and Adits should be written off under the heading of Depreciation each year for its expected life. Development up to the Revenue stage may be placed to a Temporary Development Account, or may be charged to Capital Account finally. After the producing stage is reached, all expenses in connection with deepening main shafts, cutting stations, and development should be charged direct to Working Costs, or to a Development Redemption Account. In the latter case every ton treated is debited with its proportion of redemption. No expenditure should be debited to Capital Account, except for large special items, e.g. purchase of additional property, increase of machinery, plant, or buildings (but not merely the replacement of them when worn out), sinking of new main shafts, excessive development of payable ore. All such items should bear their proportion of administration and general expenses, but not repairs and maintenance.

If an old shaft or mill is replaced by a new one, the old mill or shaft should be written off from Capital Account against Revenue, either at once or by instalments; if this is not done, they should be charged direct to Revenue, if necessary by instalments, so as to spread the expense over a longer period, and so equalize matters. Depreciation, when written off Profit and Loss Account, only accumulates cash for future shareholders, which reduces the rate of dividends for existing shareholders. Its main object is to make adequate provision out of Revenue for such new machinery, plant, etc., as are almost certain to be required before the mine is worked out, and incidentally it reduces the Income Tax and Fire Insurance Premiums. Instead of deducting depreciation, some prefer to make a small charge on every ton of ore treated, or they appropriate every year a certain proportion of the profits for a Reserve Fund. The gross or net value of unfinished products, e.g. ore at grass, should not be considered as an asset, but when taken into account should be credited only with the cost, provided the cost does not exceed the market value; but bullion, concentrates, blister copper, and other marketable products ready for shipment, should be shown as assets in the Balance Sheet, and included as Revenue, after deducting all possible returning and other realization charges.

The assets may be classified into (1) those that are actual cash or which can immediately be converted into cash, e.g. cash in hand or balance at bankers; (2) assets which can be realized at comparatively short notice, e.g. investments that can be sold readily on the Stock Exchange; (3) assets which will take a certain time to realize, e.g. investments not quoted on the Exchange or which must be realized gradually, also amounts due from debtors; (4) doubtful assets which are only expected to realize a certain percentage of their value, e.g. machinery and buildings, doubtful debts—when debts are known to be irrecoverable they should be written off; (5) other items which can only be realized as long as the business is a going concern, e.g. the amount paid for the property; (6) unrealizable items, the inclusion of which may be quite proper so long as the mining company exists. Some companies have had to go into liquidation in spite of a surplus of assets over liabilities, in consequence of their resources being locked up. If a reserve fund has been created, this only means that a provision has been made from a book-keeping point of view for a possible deficiency or shrinkage in value of items appearing on the credit side of the Balance Sheet, and this amount should be represented by investments outside the business. When the balance brought forward from the Profit and Loss Account is on the credit side of the Balance Sheet it shows the concern has been working at a loss. A comparison with former Balance Sheets should show whether such a loss has been persistent, or whether the profits and losses have fluctuated.

The grouped balances on the debit or liabilities side of the Balance Sheet often include items that are not strictly liabilities. On this side are set out the liabilities of the Company, including not only outside liabilities or debts due to creditors, but also the indebtedness of the concern itself to sundry impersonal accounts, including a surplus or undistributed balance standing to the credit of the Profit and Loss Account. It is usual to head the debit side with details in connection with the capital. The capital may consist of *ordinary shares* which bear the brunt of risks, and, in consequence, should be entitled to receive the larger share of the profits; *preference shares* which have the advantage of being entitled to a first claim on the profits of the undertaking, but usually to a limited extent; such shares may be given to vendors in part payment, or may be created when fresh capital is required. Preference shares may only be entitled to participate in profits earned in each particular year, without the right to have any deficiency made good out of the profits of succeeding years, or the shares may be known as *cumulative preference shares*, which entitle their holders to the full amount of their minimum dividends being satisfied out of future profits, no matter how much such dividends may be in arrear, before the owners of ordinary shares receive any return on their capital. The shares may be preferential for dividend-receiving purposes, but may or may not be preferential with regard to the distribution of assets when the Company is wound up. Money may be borrowed on debentures secured by a mortgage. Power is generally given to buy up a certain number of debentures a year till they are all redeemed; in the meanwhile an agreed interest has to be paid on those outstanding. A shareholder should know the terms in connection with preference shares and debentures, and he should be able to ascertain from the Balance Sheet whether and to what extent any of the creditors are secured, and what portion of the property has been hypothecated. Paper or nominal capital is never decreased, unless it is considered too unwieldy or it is desired to evade taxation. Its main care is to let the shareholder know what proportion he holds in a mine. The actual value is indicated by the market price of shares, which should vary with the conditions of the mine. The shareholder should consider whether the capital is sufficient to open up and equip the mine properly. The nominal capital is not always fully subscribed, and the shareholder should note whether more shares have been subscribed for since the last period, as it is on the distributed shares that dividends have to be paid.

The formation of the company should be carefully investigated. If the vendors have been paid heavy premiums this may prevent the mine from paying adequate dividends to subscribing shareholders. It is a mistake to have too many shares at first; if they rise in value they can always be divided so as to be less cumbersome for ordinary investors to handle. If a mine does not float well, and the shares are left in a few hands, there is always a chance that large parcels may be forced on the market at one time. A large amount of capital does not imply security. If too little capital is provided the company gets into debt, and has to create new shares (generally preference shares to encourage outsiders to subscribe to what has so far been a failure) or money has to be raised on debentures, so the original shareholders may have to wait a long time for interest on their capital, if indeed they ever receive it. Vendors are sometimes given paid-up shares in part payment, and this is often put forward to show the faith the vendors have in the property. In most cases they have no choice, so make a virtue of necessity, but if they accept shares in lieu of cash they generally demand shares of a larger face value than the cash they were prepared to take, as in many cases they sell below par and incidentally bring down the value of the subscribed shares. Anyhow the paid-up shares created absorb a certain amount of profit, lessening the dividends for the subscribed shares. The balance of Profit and Loss Account when carried to the debit side of the Balance Sheet shows the amount available for distribution. The sum of this and other items on the debit side must correspond with the sum of the items on the credit side, but the apparent balance may be fictitious if the proper proportion has not been written off depreciating assets.

The Profit and Loss Account, or as it is sometimes called the "Revenue Account," is a statement of income and expenditure. The income side should show the sale of metals, etc., and other items of income grouped under suitable headings, e.g. rents of cottages, discounts, rebates, interest, etc., and a comparison should be made with the Profit and

Loss Accounts of the preceding few years. The expenditure side should show the various items of expenditure under their proper headings, e.g. office expenses (including directors' and auditors' fees), agency expenses, advertising and stationery, postages, telegrams, and petty cash, general expenses, legal expenses, cables, travelling expenses, rent and other charges in connection with mining leases, exchange and bank charges, land-tax, insurances, interest, donations to hospital, mining account, smelting account, freight, realization charges, dividend and income taxes, depreciation (written off), etc., and should include all unpaid accounts owing at the end of the period, the balance (if any) being profit carried to Balance Sheet.

Sometimes the Profit and Loss Account is split up, a working account being shown which is debited with such items as mining, ore treatment, and freight, and credited with value received from products. The balance is then carried to the Profit and Loss Account, which is debited with such items as office expenses, rent, taxes, general expenses, etc., the balance being gross profit or loss. If the former, this is carried over to the credit side of a subdivision of the Profit and Loss Account, and added to any credit balance over from the previous year. On the debit side is shown how the money has been distributed, e.g. in depreciation, purchase of debentures, interest on debentures, etc., the balance being net profit carried to the Balance Sheet and available, if sufficient, for the payment of dividends.

The main subdivisions of the ordinary Profit and Loss Account should correspond with the main subdivisions of the cost sheet, e.g. (1) development, (2) ore extraction, (3) sorting at surface, preliminary crushing, and transport, (4) ore treatment (dressing, amalgamation, smelting, etc.), (5) administration charges and general charges at mine, (6) realization charges on products, (7) taxes and royalties of all kinds, shown separately, (8) head office charges.

Cost keeping gives an intimate knowledge of the inner working of a concern. The object is to obtain the total cost of work and an analysis of such cost. Intelligently used it enables a manager to detect waste of material, loss of time, in fact to lay his finger on leakages, so that he can reduce his expenditure and increase his profits. Costs are useful for future reference, and serve as a check on similar work. Cost keeping is a system of recording the materials used and the labour employed on a particular job. To draw up a system suitable for any particular mine, requires a man with practical experience of the requirements of mining and methods employed. The system of cost keeping need not be complicated, but the smaller the company the less elaborate it should be. The fact that a company has a large capital is no excuse for extravagance. It is generally found that when once the mine officials have the object and utility of costing properly explained to them they take a great interest in it; and their co-operation is necessary for the success of the scheme.

The system should be made as automatic as possible by getting others to fill in the forms as far as they can. There should be no repetition necessitating copying, except totals; any duplicates required are done with carbon paper. So far as possible everything is worked out on the double entry system, which serves as a check against fraud and personal error. Technical results are expressed in ounces, per cent, tons, or feet, and should not be confused with commercial results expressed in £ s. d. The card system is used where applicable, as being time and labour saving. If no proper accounts are kept, material may be used on a job and never be entered up. Except during construction when requirements are abnormal, the storekeeper has the best idea of what stock should be kept in hand to suit local requirements, taking time for transit into consideration, rainy seasons, droughts, etc. The various foremen know beforehand if any special stock is likely to be required, and may notify the storekeeper personally or on a special form. A list of stores likely to be required is often printed, and tenders called for supplies. From the tenders the storekeeper can ascertain the current prices for, say, a year.

When any stock gets low the storekeeper fills out an order form, stating the size, quantity, and quality of the article required. A carbon copy is made to be filed in the office and the price entered on it. The original is sent to the merchant and to it is attached the Company's duplicate invoice form; some of the blanks on it are filled in before the form leaves the office, others on its return, while the merchant fills in the other spaces. This serves the purpose of a Stores Inward Book. The order number in the consecutive number of the order as sent out, commencing afresh every month; the storekeeper's progressive number also starts afresh every month, and is the consecutive number of invoice as received back. The sea freight is obtained from the agent, the railage from the railway company, but is of course checked as to rates. The total is then transferred to Stock Cards. The merchant returns the Company's duplicate invoice, together with his original invoice. The duplicate is filed at the mine, while the original is sent to the head office in support of vouchers for payment. There is a separate Stock Card for each kind or size of article, and they are printed on each side. The stock received is filled in from the duplicate invoice. What has been issued can be traced through the number of the Requisition form; the balance in stock can be seen at a glance, while the minimum stock that should be kept in hand is noted in one corner.

Stores Requisition forms are printed in books with butts, and should be of such a size that they can easily fit in the foreman's pocket. They are worded in such a way that they can be filled in for any department. The form is filled in and initialed by a foreman, and when an article is given out it is signed for by the employee receiving it.

Job cards are printed in two colours, one for surface jobs, the other for underground. Each job has its special number. A fresh card is used every month for each job. The total of the card is posted to its account in the Ledger, and the daily expenditure to the Daily Cost Sheet. Each man has a token with the number assigned to him on it. Before proceeding to work he hands this to the timekeeper who gives him a Labour Daily Time Check in exchange; this may be printed on different-coloured paper for the different shifts. The employee fills in the check with his name, the class of work he has been employed on, and the time occupied at the different jobs; this is signed by his foreman and handed to the timekeeper at the end of his day's work. The next day the timekeeper apportions the labour to the different jobs on which it has been employed and enters it on the Labour Job Cards, which, like the Stores Job Cards, are also in different colours for surface and underground jobs.

Cost Sheets are prepared periodically, say monthly, for the benefit of the Management and Directors (not for publication), in order to show the progress of business. All items which occur regularly should appear under the same heading each time. Suitable main headings are—

(*A*) *Mining.* (1) Mine Capital Expenditure (shaft sinking, plat cutting, and shaft bins); (2) Mine Development (shaft sinking, plat cutting, driving, cross-cutting, rising, winzing, diamond drilling); (3) Ore Extraction (breaking ore, shovelling, filling stopes, trucking, hoisting ore).

In connection with these, details should be given of gross figures for the following, as well as the cost per ton: (*a*) Shifts worked (by contract, by day labour, (*b*) Work done (footage sunk, risen, or driven, or tons broken), (*c*) Wages (miners, truckers, timbermen), (*d*) Management and General Office Expenses, (*e*) Stores (showing quantity and value of explosives, candles, timber, general), (*f*) Power (hoisting, drills, pumping), (*g*) General Charges, (*h*) Sampling, Assaying, and Surveying, (*i*) Total Expenses for the month, (*j*) Total Expenses to the end of previous month, (*k*) Total Expenses to date.

(*B*) *Sorting at Surface, Preliminary Crushing and Transport.* This is intermediate work between the mine and the treatment works, and may have sub-headings, e.g. (*a*) Labour (pickers, breakers, truckers), (*b*) Management and General Office Expenses, (*c*) Stores, (*d*) Power, (*e*) General Charges, (*f*) Total Cost, (*g*) Gross Tonnage handled, (*h*) Proportion of different Classes of Ore sorted and Waste discarded.

(*C*) *Ore Treatment.* This naturally varies with the class of treatment, whether ore dressing, amalgamating, cyaniding, smelting, etc., and whether one or more methods are employed. Each department should have its own special detailed Cost Sheet; for instance, a gold-mine might have the following sub-headings: (1) Milling, (2) Concentrating, (3) Roasting, (4) Fine Grinding, (5) Cyaniding, (6) Filter Pressing, (7) Precipitation and Smelting, (8) Disposal of Residues, giving items, e.g. (*a*) the Quantities treated, (*b*) Labour, (*c*) Salaries, (*d*) Stores, (*e*) Repairs and Maintenance, (*f*) Power, (*g*) Water, (*h*) Assaying and Sampling, (*i*) Various Supplies (quicksilver, shoes, and dies, cyanide, lime, zinc, sulphuric acid, filter cloth, fluxes, coke, etc., giving their quantities and values). A smelting proposition, depending on its nature, may have headings such as Calcining, Smelting, Converting, Refining, etc., and the Various Supplies would contain such items as water-jackets, slag-pots, etc.

(*D*) *Management and General Office Expenses.* This may have sub-headings such as (1) Consulting Engineer's and General Manager's Fees, (2) Office Staff, (3) Stationery, Postage, and Telegrams, (4) Medical and Sanitary Charges, (5) Travelling Expenses, (6) Fire and Accident Insurance, (7) Stabling, (8) Local Bank Charges. These are charged against different Departments in proportion to the costs of each Department.

(*E*) *Realization Charges on Products*, with sub-headings of (1) Transport to Railway, (2) Railway Freight, (3) Shipping Charges, Agency, and Commission, (4) Sea Freight, Insurance, etc., (5) Selling Expenses, (6) Returning Charges, etc.

(*F*) *Income Tax, Royalties, and other Taxes.* Head office charges, e.g. rent, directors' fees, legal expenses, bank charges, auditors' fees, foreign agency expenses, advertising, and interest on loan or debentures, are not known at the mine, so cannot be added.

Separate accounts should be shown, giving on the debit side the cost, and on the credit side how it has been distributed to the main headings for Power, Hoisting, Compressed Air, Pumping, Sampling, Assaying and Surveying, Repairs and Maintenance, General Charges (blacksmith, foremen, etc.), and if Construction and Equipment is being proceeded with, show sub-headings (1) Machinery and Plant, (2) Buildings, giving the cost of any job to the end of the previous month, the cost for the present month of labour, machinery, timber, stores, management, and general office expenses, etc., and the total expenditure to date. A Distribution form shows the total cost of wages and stores for Mining and Treatment, Extraction, and other headings. A Stores Account shows the value of stores on hand at end of the previous month, what has been received during the month, issued during the month, and on hand at date. A Cash Account shows the balance at Bank at the commencement of the month, the money received from head office, sundry receipts, expenditure for the month, and balance in Bank at date. A Summary shows the Expenditure to end of previous month, Expenditure for present month, Total Expenditure to end of the period for (*a*) development, (*b*) mining, (*c*) ore treatment, (*d*) construction and equipment, (*e*) general expenses, (*f*) realization on products, (*g*) total ordinary working costs.

Technical details according to the class of treatment should have in tabulated form the number of hours run, tons of ore treated (from the mine, purchased ore), tons of revert (first matte, converter slag, flue dust, etc.), fluxes (ironstone and limestone), weight of products, value of metals, yield per ton, percentage of extraction, ore on hand, etc. The Capital Account shows Development (after deducting charges to ore extraction), Construction, and Equipment, Total Expenditure, Less Sales of Plant, Net total of Capital Account. The Working Account shows Ore Extraction (including charges on Ore from Development), Treatment, General Expenses, Realization, Total Ordinary Working Costs, Retreatment, Grand Total Working Account, Gross Income from all Sources, Net excess (or loss) over working expenditure, Net excess (or loss) over all expenditure. The tonnage of broken ore in the stopes at date and its cost per ton.—(N. Barker).

SURVEYING

The True Meridian

There are two meridians: the True, which is permanent, and the Magnetic, which is variable. The angle between the magnetic north and the true north is known as its declination.

A rough determination of the true north can be made by the following methods—

Shadow Method. This is based on the fact that at equal distances of its passage across the meridian the sun is at equal altitudes above the horizon: consequently at these times it gives shadows of equal lengths. One or more concentric circles are made on a board. In the centre is placed a vertical rod. A little before and after midday note where the ends of the shadows come in contact with the circle. Connect where the shadows end at a circle. Bisect this line and join it to the centre of the circle. This represents the true meridian.

The Method of Corresponding Altitudes. This depends on the fact that a star when at equal altitudes above the horizon of a place is equally distant from the plane of its meridian. The *A* vernier of a theodolite is set at zero, and the *B* vernier reading is also noted. The limb is then moved horizontally, and the telescope vertically so as to bring it to bear on a fixed star of the first or second magnitude, some two or three hours before its culmination. As soon as the star is intersected on the cross wires, both the horizontal and vertical circles of the instrument are clamped and the reading noted. After its culmination the star will again, later on, have the same altitude as it had when first read. The telescope is kept clamped and the theodolite moved horizontally till once more the star is intersected by the cross hairs. The horizontal circle is then clamped and both verniers read. Half the sum of the two readings on the horizontal circle will represent the true meridian.

The Pole Star Method. In the northern hemisphere one may obtain the true meridian by noting when the Pole star is in the same vertical plane as the star Alioth (E Ursae Majoris), which may be done by watching for the moment when a suspended plumb bob will cover both stars. The Pole star is exactly in the meridian 29 min after it has been in this vertical plane.

To Connect Underground Workings with the Surface or Transfer the Azimuth Underground

By Means of an Adit. The traverse may be continued direct from the surface.

By Means of an Inclined Shaft. The traverse on the surface is continued down the shaft to the first level if possible; otherwise as far as can be seen, where an intermediate station must be set up. The angle of depression is read as well as the horizontal angle. From the incline distance and the angle of depression, the vertical depth can be calculated.

Transferring the Azimuth Down One Shaft. Two wires of steel or brass (if there is much iron about, brass should be used), the diameter of which depends on the weight of the plumb bob used, are let down the shaft, free from all obstructions. While lowering the wires, plumb bobs weighing 8 to 10 lb each are attached, but when the wires are as far down as desired, these are changed for heavier bobs weighing 50 lb. Each of these bobs is submerged in a vessel containing water or oil to steady it, which must be protected from water falling into it for fear of disturbing the bob. A theodolite is set up and adjusted 12 to 20 ft from the nearest wire so as to be in line with the two wires. The vernier plate is set at zero, the telescope directed to a known back sight, and the horizontal angle read. This is repeated several times, and if the readings are sufficiently close, their mean is accepted. The distance of the instrument to each wire, the distance between the two wires, and the distance between the instrument and the back sight are carefully measured. Underground the

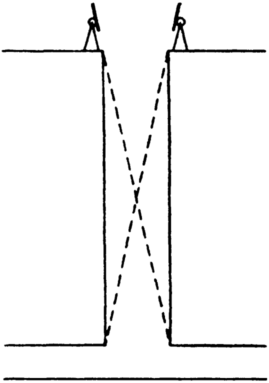


FIG. 68

instrument is set up as at the surface 12 to 20 ft from the nearest wire and brought directly in line with both of them. It is then directed to a station placed at a convenient distance for sighting, and the reading repeated several times. Similar measurements are made to those on the surface.

Connecting by Means of a Transit. The instrument used has an eccentric telescope, either superimposed or on one side. A point on either side of the shaft is fixed from the surface survey, and from these points sights are taken to lights below at opposite sides of the shaft. Where auxiliary telescopes are used, corrections have to be made either to the horizontal or vertical angles according to the position of the auxiliary telescope.

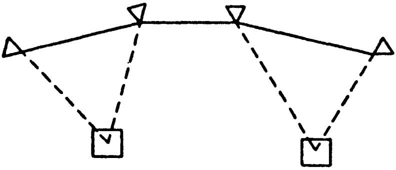


FIG. 69

By Means of Two Shafts. A wire is let down each shaft in a similar way to that already mentioned. Make the wires the apexes of triangles, the bases of which are suitable triangulation lines. By measuring the angles of the base to the wires, the position of the wires can be determined. Underground a theodolite is set up at a convenient station previously connected with the remainder of the underground traverse and the position of the plumb line fixed with regard to the underground survey. This is done at each shaft.

How to Set a Map

A map is said to be *set* when it is laid out so as to correspond with the ground.

With Compass. If the magnetic north is shown on the map, lay the compass over it and turn the map round slowly without disturbing the compass until the north end of the needle corresponds with the magnetic north of the map. If the true north only is shown, and you know the magnetic variation, mark off the variation with a protractor and proceed as above.

By Objects. Find your position on the map, also identify on the map some distant object you can see. Join these two on the map with a straight line, then turn the map about the point marking your position till this line points to the distant object.

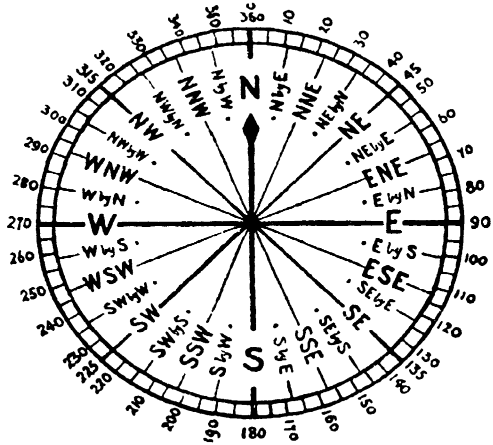


FIG. 70. THE COMPASS

There are 32 points of the compass. The names of the points are as follows—

1st Quarter N.	2nd Quarter E.	3rd Quarter S.	4th Quarter W.
N. by E.	E. by S.	S. by W.	W. by N.
N.N.E.	E.S.E.	S.S.W.	W.N.W.
N.E. by N.	S.E. by E.	S.W. by S.	N.W. by W.
N.E.	S.E.	S.W.	N.W.
N.E. by E.	S.E. by S.	S.W. by W.	N.W. by N.
E.N.E.	S.S.E.	W.S.W.	N.N.W.
E. by N.	S. by E.	W. by S.	N. by W.
E.	S.	W.	N.

The Plane-table

It sometimes happens that a mining engineer is unable to obtain a map of the district he is called upon to examine, and he has neither time nor proper instruments with which to make a survey. In such a case a 14 in. square plane-table would be most useful, but if that is not available he can make shift with a board to which paper is affixed, a four-folding two-foot rule, and a compass.

The surveyor sets up his board at No. 1 station and notes the magnetic north. He then sights on to No. 2 station by placing the 2 ft rule on the board with the two ends turned up at right angles to serve as sights, and a line drawn in its direction. Sights are now taken to other prominent points, and lines drawn towards them. The distance between Nos. 1 and 2 stations is measured on a scale not less than 6 in. to a mile. On removing the board to station No. 2 he sights back to No. 1 to obtain his proper position, and then sights and draws lines to the other stations. Where the lines from the two stations meet there is the position of the other stations, on the principle that if one knows two angles and one side of a triangle, one can find the other angle and two sides. The plan of the survey is drawn in the field: no calculations are required and details are readily sketched in.

Transit Theodolite

When taking the instrument out of its box, lift it by placing the hands beneath the levelling base. When about to move the instrument on its tripod from one place to another, see that the levelling screws are bearing properly, the needle is lifted, and all clamps are sufficiently firm not to allow any motion in case of a slight shock.

Certain adjustments have to be made from time to time. If the instrument is badly out of adjustment, it is better to bring it gradually into adjustment as a whole than to attempt to adjust completely one part at a time. The adjustments may be temporary, as when levelling the instrument at every station; or permanent, which should be tested at frequent intervals, and if necessary be adjusted periodically.

To adjust the plate levels, fix the instrument rigidly on its tripod, then rotate the horizontal circle until the telescope is parallel with a diagonally opposite pair of plate levelling screws. Bring the bubble of each plate level to the centre of its run by means of the foot screws turning the opposite screws of each set simultaneously at the same rate in opposite directions; then rotate the instrument until the telescope is parallel with the other pair of diagonal foot screws, and again bring each bubble to the centre of its run by means of the foot screws. The plate bubbles should then be level throughout a complete revolution. If not, correct half the difference by the foot screws, and half by the capstan-headed screws on the plate level. Repeat if necessary.

See that the cross-hairs are truly vertical and horizontal. Sight on to a plumb-bob line. Although the maker takes care that the horizontal and vertical hairs are at right angles, the rim in which they are placed may become slightly shifted. If this is the case, slightly loosen two adjacent capstan screws and tap their heads gently with some small instrument, such as a knife, so as to turn the ring a little until the hairs become truly vertical: then tighten the screws again.

Screw the object glass firmly home and then make a short continuous scratch on the ring of the glass and upon its slide so as to be able to see at any time if the glass is in the same position, otherwise it may materially derange the adjustments.

"Adjust the line of collimation, that is, placing the line of sight accurately at right angles to the horizontal axis. Level the instrument; direct the telescope to any well-defined object as far distant as possible; obtain an accurate intersection of the cross-hairs on the object. Then carefully lift the telescope out of its bearings and replace with the ends reversed. Revolve vertically and again sight the telescope to the same object. If the cross-hairs accurately intersect it, the line of collimation is perpendicular to the horizontal axis; if not, a correction must be made, one-half by the tangent screw of the vernier plate and half by the capstan-headed screws controlling the diaphragm, until the intersection of the cross-hairs and the object coincide. Then once more reverse the telescope, and if necessary repeat the operation until the adjustment is correct.

"To adjust the telescope level so that it will be parallel to the adjusted line of collimation in azimuth, sight a distant clearly defined object at a considerable elevation or depression; reverse the instrument, rotate the telescope vertically and sight the same object, and again take the reading of elevation or depression. If the two readings do not coincide, set the verniers on the mean of the two angles and bring the cross-hairs on to the point by means of the screws which keep the index in its horizontal position. The level must then be corrected by means of its own adjusting screws.

"To ascertain whether the horizontal axis is perpendicular to the vertical axis, that is, to so adjust the standard that the instrument will transit on a perfectly 'plumb' line. Fix the intersection of the cross-hairs exactly on a clearly-defined object at a considerable elevation, such as a church spire: then direct the telescope by turning the vertical limb (without moving the tangent screws on the vernier plate or the bottom plate) until the cross-hairs cut some other well-defined point near the ground and as far below the first point as possible. Turn the instrument through 180°, transit the telescope, direct again to the higher point; now depress the telescope, and if the lower point is again intersected by the cross-hairs the transit axis is horizontal, the line of collimation passes through the object, and the adjustment is correct. If the cross-hairs do not intersect at the lower point, correct half the error by the tangent screw of the supports, repeating the operation until the desired adjustment is obtained."—(T. G. Hanton.)

Tapes are made of steel or bronze. Those of 50 to 100 ft are usually thin steel ribbons; when 200, 300 or 500 ft they are made thicker to give strength, and narrower to reduce the weight. The heavier tapes have a handle at each end. In hot workings corrections must be made for expansion of the metal. Bronze tapes are used in wet and copper mines.

Surveying Stopes. This is mostly carried out in order to determine the amount of ore broken on contract, or the amount of filling done on contract, and this generally means the measurements have to be made within a short time. There are various methods employed depending on the size, regularity or irregularity of the deposit, its inclination and the system of stoping employed. The datum point for the height of a stope is generally taken from the top of a rail in the level below at a pass. The instruments used are a 10 ft levelling staff, a tape measure, and a Brunton's pocket transit or a prismatic compass, since an ordinary mine transit would be too awkward and slow under such circumstances. The volume is obtained in cubic feet. To convert these into fathoms divide by 216. If desired to convert cubic feet into tons, a factor must be determined for the particular ore, since some are heavier than others.

The Aneroid Barometer. This is only approximate, and should only be used for short intervals as it is affected by changes in temperature and atmospheric pressure; still it is useful for making preliminary levels for water race surveys. The best way is to use two barometers. One is kept at the station and is read at regular intervals if out of sight of the man with the travelling instrument. This allows corrections to be made for temperature and atmospheric pressure. A thermometer should be read at the same time. If only one instrument is available, it should be read again at the initial station as soon as possible to obtain the correction. When reading the barometer it is best to hold it vertically by its ring, not in a hot hand, and to tap the front gently in case the hand has stuck.

The Vernier

The vernier is a contrivance whereby the intervals between the divisions of the primary scale can be accurately measured. It is a scale where the length is generally one unit less than a certain number on the primary scale. Supposing the primary scale is divided into degrees and half degrees (30′), we take the length equal to 29 of the subdivisions and divide this into 30 equal parts. Where a line on the primary scale corresponds with a line on the vernier, we read the number of minutes, to which must be added, if past a subdivision on the primary scale, 30 minutes. In the illustration the lower line reads 50 deg 30 min, and the vernier reads 7 mins, making the angle 50 deg 37 min.

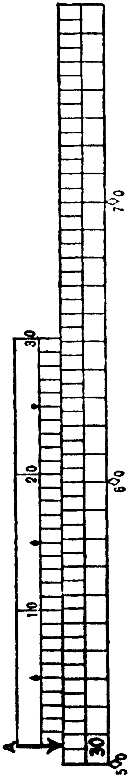


FIG. 71
THE VERNIER

Closing a Traverse

Angular Error. In closing the traverse of an area, the allowable angular error will depend on the nature of the survey and its purpose. As a rule the error should not exceed $X\sqrt{N}$, where X is the minute or fraction of a minute to which the instrument will read; and N the number of rigs. With a 5 in. theodolite, the limit of error per station would be 25 sec. In a closed traverse with 9 sides, the error would be $25\sqrt{9} = 1$ min 15 sec. When the angular error is within these limits, it is usually evenly distributed over each angle observed. Sometimes the special conditions under which each reading was made are taken into account. In other cases, the error is taken proportionally to the size of the angle.

Linear Error. After the angular work has been done, the traverse may still not close on account of an error in chaining. The linear closing error will be equal to the square root of the sum of the square of the difference in latitude and the square of the difference in departure. The result divided by the perimeter is equal to the ratio of closure. To distribute this closing error, the best method is to multiply each length by $\frac{l}{p}$ for latitude, and by $\frac{d}{p}$ for departure, and add or subtract these products as required, where l = latitude, d = departure, and p = perimeter. The ratio of closure should not exceed $\frac{1}{300}$.—(T. G. Hanton.)

To Lay Off an Angle with a Tape

The following chart will be useful for laying off angles underground where there is no instrument available and a compass cannot be depended on because of local attraction.

Fasten the 0 end of the tape at A, then stretch the tape to B a distance of 10 ft and make fast. Next find the 30 ft mark in the tape. Carry and fasten it to A. This will leave 20 ft of slack tape between A and B.

EXAMPLE. For an angle of 50 find the 19 ft mark on the tape and draw it tight from both A and B. The 19 ft mark will make an angle of 50° at A from the line AB. For other angles draw the tape tight at the distances given on the chart. For an angle to the right use the same measurements on the right of the base line.

Bore Hole Surveying

It is important to know the position a bore hole occupies, for it does not do to assume that it goes down vertically, especially in the case of diamond drill holes. Much expense has been caused on several occasions when searching for the bottom of a bore hole which has not been surveyed, the vertical depth being shortened by lateral deviation, and the lateral position also being altered. It is well to survey such a hole, say, every fifty feet.

Several methods of bore hole surveying have been proposed and tried with more or less success, that described by H. F. Marriott (*T.I.M.M.*, Vol. xiv, p. 255) being one of the best, though the apparatus is expensive, but not so expensive as blindly seeking for the bore hole underground. In a tube of some non-magnetic metal is a chamber containing a compass. Molten paraffin is poured into this until the lower portion of the compass attachment is well below the surface. This is then allowed to solidify. When lowered into position the wax is melted by means of a current of electricity passing through a resistance coil long enough to liquefy the paraffin. When the needle has had time to assume its proper direction the current is cut off, the wax allowed to cool and lock the needle. At the surface the needle is placed in alignment with the magnetic N and S, the vertical angle being obtained from the surface of the cooled paraffin. The limit of speed for lowering the apparatus down a bore hole is 200 ft/min. Rods which are non-magnetic at the surface sometimes get highly magnetic

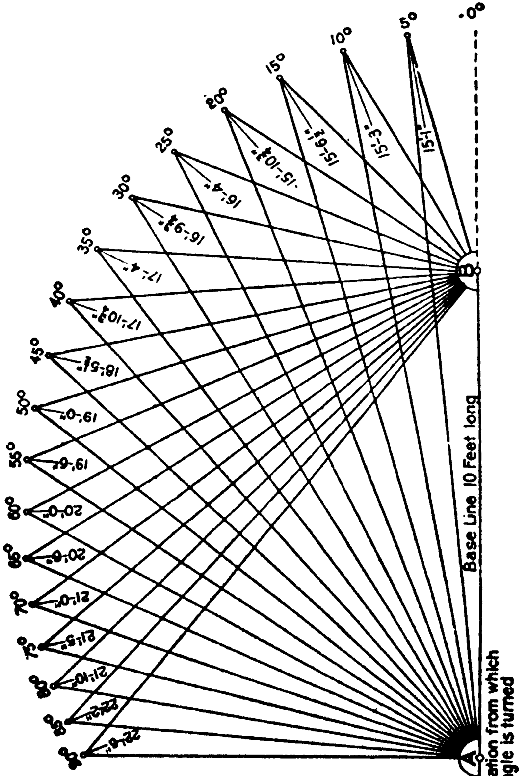


FIG. 72. LAYING OFF ANGLES

owing to friction with the walls of the hole. To prevent the needle of the compass lowered for surveying purposes from being affected, use a brass rod 10 ft long at the end of the iron rods, it being found by experiment that the deflection of the needle is unappreciable when seven feet away from a highly magnetized rod, so 10 ft should be absolutely safe. If the bore hole should pass through a magnetic stratum, one cannot help the needle being affected, but the reading of the angle of the hole will be correct, also the compass readings at the surface and sufficiently below the magnetic stratum. Thus the average of the two readings above and below the magnetic stratum would have to be noted to enable the direction the drill had taken through the magnetic stratum to be plotted.

Bore holes are not as smooth as one might think and when lowering an instrument, it is liable to receive a jar. The hole sometimes contains cavities in its side on the edge of which the instrument gets caught, and at points where the hole has been reduced in diameter there are shoulders on which the instrument may catch. It is very necessary that bore holes for tail ropes should be vertical, otherwise friction against the sides would soon wear the rope out.

Humphreys's photo-electric instrument gives a permanent record of the deflection and its direction on a single photograph, but gives a blurred picture if the clinometer and compass are not at rest.

Railway Curves (Pambour)

The outer rail of curves requires to be raised above the inner as follows, for a speed of 20 miles per hour—

500 feet radius	1·47 inches
600 ,,	0·98 ,,
700 ,,	0·63 ,,
800 ,,	0·37 ,,
900 ,,	0·16 ,,

Plans

For working plans, use medium weight paper with hot pressed surface mounted on two-ply linen which has been well shrunken. Drawings should be plotted from a scale drawn on the paper, so as to compensate for any shrinkage or expansion due to the weather.

Mine maps may be made for various purposes, and the scale to which they are drawn varies accordingly. There may be a map of the district showing the position of leases, the main ore channels, faults and other geological data drawn to a scale of 200, 400 or 500 ft to the inch. For properties with a large area, such as a colliery, the office map is generally drawn to a scale of 400 ft to an inch, and the workings to a 100 ft scale. The surface plan of a metal mine showing buildings may be on a scale of 40 ft to an inch, while the underground workings are drawn to a 20 ft scale. For underground workings it is found convenient to divide the plan into co-ordinates 50 or 100 ft apart.

In the case of plans, one set of lines run parallel with the meridian and the other set at right angles to them. One set is lettered, the other numbered. Drawings are also required for longitudinal and cross-sections. The different levels are best numbered according to their order from the surface, or according to their depth from the collar of the shaft which is taken as datum. Each level is given a distinctive colour. Cross-cuts are shown in full lines and given the colour of their level. Winzes are shown with dotted lines and are coloured grey.

An assay plan shows the width of the pay ore. If the lode is narrow, the width is written on one side and the assay value on the other. In a wide lode, the width sampled and the assay values are shown on the sides of a vertical line. Assays of the different metals in a sample are often written in different coloured inks. Lines may be drawn to enclose areas of different grades of ore to enable the manager to decide how much ore is to be taken from certain stopes to make the proper blend for treatment. If the deposit is not too steep, the different levels can be shown on the same sheet, making a composite plan. If the lode is nearly vertical, it is better to have a separate sheet for each level.

Office plans are generally drawn in pencil; but tracings are drawn in indian ink from which blue prints can be taken. The tracings should have a datum line in common, so that one sheet can be placed over another.

Blue Print Paper. Make solution (*a*) one part by weight of citrate of iron and ammonia to five parts of water. Solution (*b*) one part by weight of red prussiate of potash (recrystallized) to five parts of water. Mix equal quantities in a dark room and apply to the paper with a soft flat brush as uniformly as possible on one side only. Hang up to dry in a dark room. When fresh, the paper has a yellowish green colour; when old, it becomes dark green or bluish and will not show clear white lines. When using, expose in bright sunlight for about five minutes and then wash well in clean water. Good rag stock paper gives whiter lines than wood pulp paper.

TABLE OF ACRES REQUIRED PER MILE AND PER HUNDRED FEET FOR DIFFERENT WIDTHS

Width	Acres per mile	Acres per 100 ft	Width	Acres per mile	Acres per 100 ft	Width	Acres per mile	Acres pet 100 ft
Ft			Ft			Ft		
1	0·121	0·002	**35**	4·24	0·080	**68**	8·24	0·156
2	0·242	0·005	**36**	4·36	0·083	**69**	8·26	0·158
3	0·364	0·007	**37**	4·48	0·085	**70**	8·48	0·161
4	0·485	0·009	**38**	4·61	0·087	**71**	8·61	0·163
5	0·606	0·011	**39**	4·73	0·090	**72**	8·73	0·165
6	0·727	0·014	**40**	4·85	0·092	**73**	8·85	0·168
7	0·848	0·016	**41**	4·97	0·094	**74**	8·97	0·170
8	0·970	0·018	**41½**	5·0	0·094	**74½**	9·0	0·170
8½	1·0	0·019	**42**	5·09	0·096	**75**	9·09	0·172
9	1·09	0·021	**43**	5·21	0·099	**76**	9·21	0·174
10	1·21	0·023	**44**	5·33	0·101	**77**	9·33	0·177
11	1·33	0·025	**45**	5·45	0·103	**78**	9·45	0·179
12	1·46	0·028	**46**	5·58	0·106	**79**	9·58	0·181
13	1·58	0·030	**47**	5·70	0·108	**80**	9·70	0·184
14	1·70	0·032	**48**	5·82	0·110	**81**	9·82	0·186
15	1·82	0·034	**49**	5·94	0·112	**82**	9·94	0·188
16	1·94	0·037	**49½**	6·0	0·114	**82½**	10·0	0·189
16½	2·0	0·038	**50**	6·06	0·115	**83**	10·1	0·190
17	2·06	0·039	**51**	6·18	0·117	**84**	10·2	0·193
18	2·18	0·041	**52**	6·30	0·119	**85**	10·3	0·195
19	2·30	0·044	**53**	6·42	0·122	**86**	10·4	0·197
20	2·42	0·046	**54**	6·55	0·124	**87**	10·5	0·200
21	2·55	0·048	**55**	6·67	0·126	**88**	10·7	0·202
22	2·67	0·051	**56**	6·79	0·129	**89**	10·8	0·204
23	2·79	0·053	**57**	6·91	0·131	**90**	10·9	0·207
24	2·91	0·055	**57½**	7·0	0·133	**90½**	11·0	0·209
24½	3·0	0·057	**58**	7·03	0·133	**91**	11·0	0·209
25	3·03	0·057	**59**	7·15	0·135	**92**	11·2	0·211
26	3·15	0·060	**60**	7·27	0·138	**93**	11·3	0·213
27	3·27	0·062	**61**	7·39	0·140	**94**	11·4	0·216
28	3·39	0·064	**62**	7·52	0·142	**95**	11·5	0·218
29	3·52	0·067	**63**	7·64	0·145	**96**	11·6	0·220
30	3·64	0·069	**64**	7·76	0·147	**97**	11·8	0·223
31	3·76	0·071	**65**	7·88	0·149	**98**	11·9	0·225
32	3·88	0·073	**66**	8·0	0·151	**99**	12·0	0·227
33	4·0	0·076	**67**	8·12	0·154	**100**	12·1	0·230
34	4·12	0·078						

EXAMPLE. To make a track 12 ft wide for 2½ miles, what area must be cleared?

In the second column, opposite 12 in the first, it will be found that 1 mile occupies an area of 1·46 acres; therefore 2 miles equals 1·46 × 2 = 2·92; and ½ a mile equals 1·46 ÷ 2 = 0·73; so 2½ miles = 2·92 + 0·73 = 3·65 acres.

Conventional Signs

On the Surface

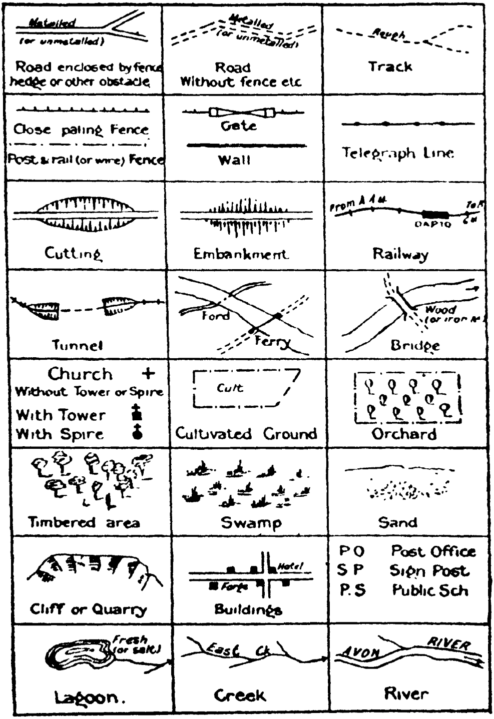


FIG. 73. ON THE SURFACE

Underground

◪◪◪	Shaft, the divisions correspond with the number of the compartments
◪	Winze top
⧄	Winze bottom
⧓	Rise top
⊠	Rise bottom
□	Chute

ORE DRESSING

General. The difference between ore dressing and metallurgy is that the former is mechanical in action while the latter is chemical. In some cases they merge one into the other, as when crushing and amalgamating; crushing gold ores with cyanide solution; or calcining to make an iron ore magnetic so that it can be separated by an electro-magnet.

A miner should know the requirements of the metallurgist with regard to the ore he produces, not only with regard to the grade, but also in connection with the associated minerals, some of which are useful, others harmful, and some only objectionable as adulterants. The miner can often assist the metallurgist by careful mining.

It is a matter of calculation whether it is cheaper to get rid of waste material by mechanical or chemical means. There is a certain amount of loss in each case, but to compensate for the loss in ore dressing, less fuel and flux are required for smelting a smaller quantity of ore, less labour, and less loss is in the slag. There is an irredeemable minimum of metal left in tailings and slags which cannot be recovered, the loss of which is dependent on the tonnage of the waste material, so one must distinguish between the contents of an ore as shown by assay and what is actually saved. Some of the losses in treatment may be temporary, while others are permanent. Temporary losses may be metal such as copper or tin absorbed in the bottoms of furnaces which is recovered from time to time; the loss of gold in rich tailings or absorbed in amalgamated copper plates; also certain fumes when condensed in flues and chambers. Permanent losses take place in tailings and slags, and by volatilization. The loss in concentrating has of course to be added to that of smelting. The known losses are only a part of the total loss. If the ore has to be transported, there is a loss in handling, wind, and occasionally by disaster and theft. If the ore is in small parcels or very rich, it is dispatched in bags. Tin concentrates are often bagged wet so as to make the material of the bags swell, though if very fine it is best put up in paper-lined bags: but pyritic concentrates must be bagged and kept dry, otherwise they will rot the bags. The bags must not be too large to handle, otherwise men will use hooks, which cause loss, and unless special arrangements are made no one will trouble to sweep up and bag what has been spilt.

The object of ore dressing is to separate the useful minerals from those which are worthless, and the former from one another. Ores that are easily converted into powder while being concentrated, or are soluble in water, are not suitable for concentrating. The increased value of an ore after dressing must at least cover the cost of treatment. The degree of concentration may depend on the cost of transport. If this is expensive it may be necessary for a copper ore to be 30 per cent, which at a later date, when conditions are improved, and smelting takes place at the mine, may be reduced to 3 per cent; or in other words the run-of-mine can be smelted direct. Again, the grade to which the ore must be dressed may depend on what is available. Though users of chrome ore can apply a lower grade than 54 per cent chromic oxide, they will not do so as long as they can obtain a higher grade.

Avoid handling as much as possible. As a rule, dressing plants are better situated at the mine than at the smelters, thus saving the expense of hauling worthless material, perhaps for miles, which might be used as filling at the mine. It is generally cheaper to pump water to the mine than to convey ore to the water.

A plant may be situated on the side of a hill, or on a flat. When on the side of a hill, advantage is taken of gravity in passing from one stage to the next. A good inclination for a dressing floor is 1 in 3. A plant on the surface has a better foundation for heavy machinery than when all the plant is enclosed in one building, but the area required, and consequently sheds, are greater. If all the machinery is under one roof, material has constantly to be elevated. In either case there must be a place where the tailings can be dumped. The crusher should be in a different building and driven by a separate engine from that of the concentrators on account of the vibration and irregular work. The mill should be built in units as the output of the mine is increased, and each fresh unit benefits from experience gained from earlier units. Should there be a shortage of ore or water, one unit can be cut out for the time being without interfering with the others.

Some machines are made to a standard size found by experience to be most suitable for general work. Such machines are cheaper than those which have to be specially designed and made, and spare parts are easily obtained. When a machine is found suitable for a certain class of work, it is advisable to keep to that machine as it saves having to stock a greater number of spare parts. A machine should be simple and strong.

In order to obtain a sale some manufacturers who do not expect to get a repeat order outbid competitors by making the machine too light or of inferior material, thus causing no end of trouble and delay, which in the long run costs more than a first-rate machine made by a reliable manufacturer. Figures given in catalogues are often only good under most favourable conditions. Although the advice of the manufacturer who has had much experience in the working of the machine he provides is of value, it is well to make inquiries from those who have used similar machines. Though one manufacturer will not openly decry the machine made by a competitor, he indirectly does so by emphasizing the advantage of his own machine.

The limits between which a machine should be run should be noted in order to obtain the necessary gearing. One should beware of obtaining an unproved machine, even if given free, especially in places where it is difficult to make alterations and repairs, for much time is lost in erecting, testing, and if unsuitable, dismantling. The designer of a plant must have a sense of proportion as to the number and average capacity of each kind of machine, the number of revolutions, and quantity of water required. In each grade of the process in ore dressing, a certain amount of waste is eliminated, consequently subsequent machines have so much less to treat. The machinery should not be cramped, for if it is, there is difficulty in making repairs or replacing a machine.

No one firm is likely to make all the most suitable machinery for a particular plant. A machine which does good work in one country may prove a failure in another country on account of the different class of labour employed. Because what is lost by one machine is saved by another is no proof that the second machine is better than the first, for if the first is duplicated and perhaps worked at a different rate, at a different angle, or with a different quantity of water, it might save just as much as the second machine. Before ordering machinery one should know about the following transport facilities—

Are the necessary means for loading and unloading available?

Are local bridges strong enough to support heavy machinery?

Can turns in the road be negotiated by long articles such as a boiler or pipes?

Should there be tunnels, whether they are large enough for bulky machinery, or whether it is necessary to sectionalize it, etc. (Aeroplane transport has now overcome many of the early difficulties of transport.)

The physical properties of minerals are made use of in ore dressing.

Hardness. Hardness must not be confused with toughness. A diamond is hard but brittle. Hardness affects the wear of crushing machinery.

Tenacity. Some minerals are tough, e.g. hornblende, talc, mica, gypsum, native copper, etc. They may be soft but are difficult to break up. A hard brittle mineral, such as quartz, will make more fines and slime with water than one which is soft and tough.

Structure. This affects the shape of particles. A mineral that breaks up into cubes like galena offers one-sixth of its surface on a plane surface. An elongated fragment such as antimonite or calcite only contacts a surface on a line. Thread-like shapes are apt to become matted, such as asbestos, and choke screens. Minerals that form scales such as mica, graphite, and talc, also choke screens. A flat grain of quartz may sink end on and contaminate concentrates. Grains that approach a sphere in form touch a plane surface at a point and are apt to roll along, such as garnets. The shape of particles has an important bearing on the power to settle in water or air.

Colour and Lustre. These greatly assist in hand picking: the yellow of copper pyrites; grey of mispickel; the vitreous lustre of quartz; resinous lustre of zincblende; pearly of talc.

Specific Gravity. This is one of the most useful means of separating minerals whether the medium is water or air. Of two particles of the same size but different sp. gr., the heavier particle will settle first. Of two particles of the same weight but different sp. gr., the heavier particle will settle first as it offers less friction to the medium. Sp. gr. also affects the momentum of particles projected in a horizontal direction as by centrifugal force when of two particles, the same size but of different sp. gr., the heavier will have the longer trajectory. Of two particles of the same trajectory, that with the smaller diameter will have the higher sp. gr.

Adhesion. Gold adheres and amalgamates with quicksilver. Diamond adheres to grease. Oil adheres to certain minerals and enables them to float.

Greasiness. This is a tendency some minerals have to repel water which enables them to float when of suitable size and shape, e.g. native copper, gold, graphite.

Electro-conductivity. Some minerals, mostly sulphides, are relatively good conductors of electricity, while others, mostly gang minerals, are not. When brought into contact with an electrode containing a static charge, the better conductors become similarly charged and are repelled like pith-balls under similar conditions, while the poorer conductors are not repelled so far.

Magnetic Susceptibility. Some minerals are notably magnetic, e.g. magnetite, some form of pyrrhotite, and iron particles from worn machinery. Others have very weak magnetism, e.g. franklenite, chromite, serpentine, garnet, dark variety of zincblende (black jack). Still others exhibit no magnetism, e.g. quartz, calcite, gypsum, and felspar.

Change to Magnetism by Heat. Certain minerals, especially those of iron, give off carbon dioxide or sulphur on heating, and become magnetic, e.g. chalybite, and pyrite.

Change of Mechanical Condition by Heat. By heating certain minerals, e.g. pyrite, some volatile ingredient is given off which lowers their sp. gr., thus enabling them to be separated from other minerals of originally similar sp. gr.

Decrepitation. Some minerals when placed on a hot plate, fly to pieces and may be separated from others which do not behave in that manner by sifting. Most barytes when heated from 200 to 300° C. decrepitate in this manner in a short time, and may be separated from non-decrepitant blende by sifting through a 40 mesh screen.

Solubility. Some soluble salts may be separated by fractional evaporation. Copper sulphate, found in the upper part of a lode or in heaps of second grade ore, can be leached out and precipitated on scrap iron as cement copper.

Of recent years the microscope has been used in connection with ore dressing. Among other things it may show that what macroscopically may appear to be uniform is really composed of two or more minerals which may affect its sp. gr. and cause locked-up particles to be carried away if treated by certain methods. It also shows the size ranges suitable for commercial crushing.

Washing

Some materials, especially alluvial, are associated with stiff clay which has to be disposed of before being further treated. There are various appliances for this purpose, the use of which largely depends on the quantity to be dealt with; from hand worked puddlers and sluice boxes, to horse or machine-driven puddlers.

The log washer, formerly made out of a hollowed log, is now replaced by a steel trough about 17 ft long which is inclined at an angle of 5 to 10° in which two parallel shafts of square section revolve at the rate of 16 to 25 r.p.m. On these are mounted a series of paddles or blades which are given a pitch of about 25° that form a spiral path round their respective shaft. These shafts revolve in opposite directions. All gearing is outside the trough. The lifting and churning action of the paddles cut up and reduce the clay. The size of the material treated varies up to 3 in. and is fed in at the lower end of the trough, being ejected at the upper end. Wash water is admitted through a long perforated pipe and overflows at the lower end of the trough carrying the clay with it. The process is continuous.

Wash trommels also work continuously. The trommel may be cylindrical or conical. If cylindrical, the axis is given an inclination of 1 in 12: if conical, the axis is horizontal, which is more convenient for driving. These trommels are 3 to 6 ft in diameter, 8 to 15 ft long, and require ¼ to 1 h.p. to work. They may revolve on shafting or friction rollers. In the latter case stirring blades may be attached to an axle which rotates independently of the drum. There are also iron ribs for raising the material. Water may play on the ore from a perforated pipe, or the trommel may revolve partly in water. The speed of wash trommels at the periphery is 1 to 2½ ft/sec, and they require 12 to 25 gal of water per min.

A common form of puddler used for alluvial which works intermittently is a ring-shaped trough 16 ft outside diameter, 4 to 5 ft wide and 3 ft deep. There is a beam pivoted in the centre; on either arm is attached a harrow consisting of a triangular or circular frame through which project iron spikes that serve to stir up the charge. The puddler is worked by a horse harnessed to a sweep. The clay is carried away by the overflow. When the gravel is cleaned, a gate situated at the side or bottom is opened through which the gravel and sand pass to sluice boxes. Horse puddlers can treat 16 to 28 loads per 24 hours. Puddlers constructed on the same principle are made of iron, and the harrows are driven by machinery. These puddlers are about 12 ft diam. and 3 ft high. The harrows revolve 6 to 7 times per min. The stirring occupies 15 to 20 min after which the cleaned gravel passes to the sluice boxes. One machine will treat about 120 cu yd in 24 hours.

Sorting

By judicious mining much mullock may be avoided, at the same time saving the transport of worthless material to the surface.

Sorting is ore dressing by hand, and is resorted to when dealing with different classes of ore of handling size, also when an objectionable mineral such as heavy spar of a similar sp. gr. as the ore is present.

Hand sorting may take place on stationary tables, revolving tables, or picking belts, according to circumstances.

Stationary tables are used when it is necessary to break up pieces to separate different ores. Each sorter works at his own bench in front of which is a bin from which he draws the ore as he requires it. He has a block of iron on which to break the ore, and bins into which he throws the different classes.

The Revolving Table. Here boys stand round the table, and each selects a particular class of ore as the table revolves. The ore is generally sprayed with water to wash off any dust and expose the colour and structure of the different minerals.

Picking Belts. These may be of rubber or linked sheet-iron plates (steel rusts too easily) 2 to 3 ft wide, with sides turned up for 2 in. to prevent the ore falling off. Boys stand at either side of the belt which travels about 35 ft/min. In the case of ores, the sorters may pick out rich ore suitable for smelting to save it from being crushed: second grade ore to be further reduced and concentrated; and waste. Whichever is greater in bulk is left on the belt and discharged over the end. With coal it is the shale that is picked out.

Reduction

The object of reducing ore in size is either to make the pieces more convenient to handle, or to free valuable minerals from their vein stone or from each other. This may be done by weathering, heating, or by mechanical means. Weathering does not require any special apparatus but causes a loss of interest on the capital locked up. Diamonds when encased in their matrix are not likely to be damaged when freed by weathering as they would be if broken up by ordinary mechanical means. Quartz heated in kilns is made more brittle and can be broken up more easily, yielding about 15 per cent more crushed stone with the same power. But other things have to be considered. If any pyrites present is not completely oxidized, the sulphate formed may be harmful to subsequent treatment; and any charcoal that gets mixed with a gold ore is detrimental to treatment by cyanide solution, it being a precipitant for that metal.

Methods of mechanical reduction may be classified as—

1. Crushing, by jaw breakers.
2. Squeezing, as with rolls.
3. Percussion, as with stamps.
4. Grinding, as with pans.
5. Abrasion, as with pulverizors.

Each class of reducing machine has its own sphere of usefulness, though they may overlap: and there are several machines in use for the same class of work.

Crushing

Jaw Breakers. These have a knapping action somewhat like a hammer which breaks the stone along its line of least resistance. They are employed for breaking up coarse material, and in so doing do not make an undue proportion of fines. Jaw breakers are measured by the length and breadth of the feed opening. If the feed is less than 2 in., it is better to use some other type of crusher.

There are three types of jaw breakers: the Blake, the Dodge, and the roll. They all have two jaws; one stationary, the other swinging. The swinging jaw of the Blake type is pivoted from above; the Dodge from below, and the roll in the middle. The Blake type has the jaws faced with plates having vertical angular corrugations which can be replaced when worn. The stationary jaw is held in place by check plates on each side and is bedded in zinc ¼ in. thick directly against the frame. The angle between the jaws is generally 20 to 25°. The pulley revolves 225 to 250 times a minute. Means are provided to prevent the machine from being damaged in the event of such a thing as a hammer head being fed in with the ore.

The Dodge type has its stationary jaw at an angle instead of vertical. The jaws are studded with bolts which wear at a different rate to the plates, so always offer a rough surface to the ore. The shaft makes 200 to 300 r.p.m. The Dodge is generally smaller than the Blake and is mostly used as a secondary crusher. The Blake crushes more ore than a Dodge of the same size, but the product is not so uniform, as the bottom opening is constantly varying in width.

The roll type has its fixed jaw at an angle; the moving jaw is curved and has a rocking motion; it does not advance or recede like the other types although it appears to do so. The jaws always remain at a fixed distance apart. This machine does not clog because the moving jaw rolls over the ore which drops out after the jaw has passed over it. All these machines can be regulated for the size they are required to make.

The power necessary to crush a given quantity of a certain rock from any known size to any other known size is directly proportional to the reduction in volume of the pieces. In other words, it takes twice as much power to crush from 1 in. to ¼ in. as from 1 in. to ½ in. cube.

Gyratory Crushers. A gyratory crusher is generally used for crushing down to 2 or 3 in. It consists of a conical hopper or bowl made of chilled iron or manganese-steel, inside of which is a heavy crushing head the shape of the frustum of a cone made of similar material to the hopper and is keyed to a vertical spindle. The bottom of the spindle is set ½ in. out of centre which gives a gyratory motion to the crushing head as it revolves. The crushing head has vertical corrugations in order to reduce slip, and consequently the proportion of fines. In the old type of gyratory, the face of the head was straight, in the new type it is curved. Adjustment for the size of the crushed material is made by raising or lowering the spindle. The crushing head revolves about 150 times per minute depending on the size of the machine. For large crushers the gyratory type is taking the place of jaw breakers as crushing takes place all the time in the same direction, there is less vibration than with jaw breakers which are intermittent, and the large hopper will hold a quantity of ore at a time. The cone or crushing head instead of being solid chilled iron may have a mantle of manganese-steel which, when worn out, is more easily replaced. The gyratory is heavier in weight and requires more horse-power than a jaw breaker, and the wear on the stationary plates is greater; but their capacity is greater, repairs are easier to make, and are less frequent.

The Symon Cone Crusher. There are two types, the Standard and Short Head. The former is for ordinary reduction, the latter for a finer product. They are both secondary crushers. The cone or conical head is gyrated by means of an eccentric.

The under-side of the head has a large spherical-shaped surface which rests on a large bronze socket bearing, and it is this bearing which carries the weight of the head and main shaft. Opposing the head is the crushing bowl which can be raised or lowered. Around the crusher is a circle of heavy coil springs so arranged for normal crushing that they hold the bowl down rigidly on the main frame: but should tramp iron or other uncrushable material enter the crushing cavity, they allow the bowl to rise with each gyration until such foreign matter has passed through.

Squeezing

Rolls. Rolls are used for secondary crushing. They consist of two cylinders mounted on parallel axes which revolve towards each other and crush the ore between them. There is very little rubbing action, and consequently but a small proportion of fines is formed. The capacity formula for rolls is—

$$\frac{T \times W \times S}{1728} = \text{cu ft/min}$$

Where T = distance in inches the rolls are set apart
W = width in inches of roll face
S = peripheral speed in inches per min
1728 is the number of cubic inches in a cubic foot.

On account of the irregularity of feed, etc., the theoretical capacity should be divided by four to get nearer the actual capacity. Rolls vary in diameter from 14 to 34 in., and in length from 9 to 24 in. If the diameter is too small, the angle formed by the rolls is too obtuse. The angle of nip depends on the diameter of the rolls, the size of the ore, and the space between the rolls. This space goes up to ¾ in. The horsepower required depends on the size of the rolls, from 1 to 12.

Ore should be crushed by degrees. Coarse rolls can take feed from 1¼ to ¾ in. and should be run slowly, about 120 ft at the periphery per minute. The next set of rolls takes ¼ in. feed and runs at 240 ft/min. If run too fast power is lost; if run too slow, the rolls are liable to stop if a hard piece gets between them which lets coarse particles through. The finishing rolls receive material from 15 to 20 mesh for reduction to 40 mesh. Three sets of rolls, 36 by 16, properly designed, with necessary screens and elevators, working continuously, should in 24 hours treat from 200 to 250 tons of ordinary ore; or 360 tons of quartz to 16 or 20 mesh with a return from screens not exceeding 10 per cent. When the percentage of ore remaining on the screen is too high, it indicates that the face of the tyres requires resetting. The tyre of a fine roll when worn can be used on a coarse roll without having to reset it.

Roughing rolls are generally fed dry; the finishing rolls may be fed wet or dry. When wet it is to lay the dust. The core of the roll is made slightly conical. The shell or tyre is slipped over this and shrunk on or made fast by bolts. The shell must be tightened on the core from time to time, as when they are about two-thirds worn they begin to stretch. The tyres are now mostly made of manganese-steel. The face may be straight, or curved to prevent flat pieces from passing through; and may have a corrugated surface or provided with teeth. Toothed rolls are mostly used for crushing coal. Reduction in the size of feed should not exceed 4 to 1. There is not so much ore crushed by the ends of rolls as in the middle, consequently the middle wears out more quickly. To make the rolls wear more evenly, blocks of emery sometimes rest on the rolls at the ends, thus saving the necessity to dress them so often. Much time is saved by having spare rolls ready to replace those which are worn. There are several different types of rolls which have various means for preventing breakages in case tramp steel passes into the feed.

Percussion

Stamps. Gravity stamps generally work in a battery of five. The principal parts of a set of stamps are: the foundation, mortar box, with lining plates, screen, screen frame and necessary fittings, framework with stays, tie-rods and guides, stamp-stem, head, shoe, die, tappet, cam, camshaft, and driving gear, water pipes and connections. The foundatoin may be of wood or concrete. If rock is not reached at a reasonable depth, a concrete base is laid down on which the wood or concrete foundation is built. If a concrete foundation is decided on, the holding down bolts for the mortar pass through pipes built into the concrete. These pipes are a little larger than the bolts so as to allow play for careful adjustment of the space between the bolt and the inside of the pipe which is then filled up with sand; or slots may be left in the concrete for the bolts to slip into.

The framework may be of iron or wood. The former is advisable in white ant country. A platform is laid at the back of the battery at such a height that it is convenient for a man to hang up stamps when necessary. The battery posts are tied together with binders which also serve as supports for the guides of the stamps. The journal boxes for the camshaft are situated on the main posts.

The mortar box is generally cast in one piece. The form of the interior largely depends on the purpose for which the battery is required. They may be narrow or flaired; deep or shallow. It is lined with wearing plates which are easily renewed. A narrow mortar has a quicker discharge than one that is flaired. Narrow mortars are 10 to 12 in. wide; the flaired mortars are 17 to 18 in. wide and are used for crushing gold ores when amalgamated copper plates are used inside the mortar box. When inside amalgamation is carried out, one plate is fastened on the long side just below the screen, the other in a recess under the feed hole. The mortar box is 4½ to 5 ft long, and 4 to 4½ ft high. There are two covers to the mortar box, one of iron the other of wood, their object being to keep foreign matter out and splashings in. About 8 gal of water are required for each stamp per minute. It is usual to have a single discharge, but as more material is crushed than can escape, some batteries have a double discharge, one on each of the longer sides. If the ore is micaceous, or there is anything else likely to choke the screen and cause it to burst, a slit is left in the mortar box just above the screen that is usually covered with a canvas flap which can be raised occasionally for the removal of accumulated rubbish.

The bottom of the screen is 5 in. above the top of the dies so as to be clear of friction from the larger stones. The screen frames rest on wooden chuck blocks of various heights so that the frames can be lowered as the dies wear down. To retain the same height of drop, the tappet is moved up on the stem for the required height. In order to keep the stamp the same weight as the shoe wears down, a compensation weight made in two pieces is clamped to the stem. One can tell by the sound from a distance if the stamps are being properly fed. If hand-fed, the feed platform should be level with the feed hole of the mortar so that the ore can be shovelled in instead of having to lift every pound. The feed hole is 2½ in. wide.

Steel screens are best for quartz, but are liable to be destroyed if copper sulphate is present in the ore, and if out of use for a time they become rusted. When copper sulphate is present, brass wire-cloth lasts longer, but brass is no good if quicksilver is used in the mortar box. Holes in sheet metal may be round or slotted, arranged horizontally or diagonally. If the holes have a burr, the burred side is placed inside the mortar box. The size of holes in punched sheet is given as the number of needle which just fits it.

No. of Needle	Corresponding Mesh	No. of Needle	Corresponding Mesh
5	20	9	40
6	25	10	45
7	30	11	50
8	35	12	60

If no screen is used, the height of the discharge regulates the size of the grains. The screen frame may be arranged vertically or at an angle of 10 to 12° to facilitate discharge and reduce wear of the screen. The life of a screen may be lengthened by reversing it in due time.

The dies should be a little larger than the shoes to allow for slight play in the guides. The shoes should be a little harder than the dies. If the shoe is of chrome-steel, the die should be of forged steel. The dies may be cylindrical on a square base with the corners cut off for easy removal, or octagonal. The octagonal have the advantage that when partly worn they can be reversed. A die will last about twice as long as a shoe, as it is partly protected by a layer of ore.

The arrangement of cams on the camshaft regulates the order in which the stamps fall, which is important, for if they fall in regular order, the feed would tend to pack up at one end; moreover, the order of drop affects the wear of the dies for all the stamps are not doing the same amount of work. Counting from the end nearest the driving pulley, the order known as the Homestake drop in which no stamp follows another consecutively, 1, 4, 2, 5, 3. In the Californian or scroll drop two stamps drop in succession, 1, 5, 2, 4, 3, or 1, 4, 2, 3, 5. If ten stamps are worked from one camshaft, the order may be 1, 6, 5, 10, 2, 7, 4, 9, 3, 8. Each stamp revolves 1¼ to 1½ in. with each drop. No grease must be allowed on the surface of the cam, or the stem will not turn.

To increase the weight of a stamp, it is usual to have longer heads and heavier tappets, which incidentally raise the centre of gravity. To increase the diameters of the head and die would require a longer mortar box which would exaggerate the rocking tendency. There are limitations to the diameter of the camshaft, since the larger the shaft, the greater the velocity of the cam at the point of engaging the tappet, owing to the greater distance between the centre of the camshaft and the centre of the stem. Cams are generally two-armed, the face being 2 to 2½ in. wide which strike the tappet half an inch from the stem. The rate of drop must not be so great that the cam may strike the tappet before the stamp has completed its fall; also if too fast there is so much commotion in the mortar box that pieces to be crushed may not have time to rest on the die. Heavy stamps give from 80 to 90 drops per minute. Stamps may be given a slow and high drop, or quick and short drop. A slow drop gives the material more time to settle and a better chance to be crushed finer. The end or toe of the cam is modified so as to allow the tappet to pass it at the end of a lift without being damaged.

The stem is of soft steel 10 to 15 ft long, and 3 to 3½ in. diameter. It is generally tapered at each end for 6 to 8 in. for insertion into the head should the stem break, which it generally does near the head. Should such an accident happen the stem should be annealed. The stem sometimes has a hole near the top to enable a block and tackle to be attached for raising and lowering the stem when necessary to make repairs. The stem is supported by guides, one placed about 3 ft above the cover of the mortar box, the other about a foot below the top of the stem when at rest on the die. The bearings must be long.

The tappet or disc is a cylinder of steel which passes round the stem. The face of the tappet next the stem has an annular ring ½ in. wide because the cam does not come within that distance of the stem; if this were not done, a rim would be left when the rest of the tappet was worn down.

The head or boss is cylindrical in shape, made of steel and of the same diameter as the shoe. It serves as a connection between the shoe and the stem and adds considerably to the weight of the stamp. There is a conical socket at either end, one for the stem to fit into, the other for the neck or shank of the shoe. A channel is cut through the head at the base of each socket to enable the stem and shoe to be loosened when necessary to replace them.

The shoes may be of cast iron, forged steel, or chrome-steel. Chrome-steel lasts 2½ times as long as cast iron, which is a consideration if freight is high. A shoe weighs 150 to 200 lb when new and should be discarded when worn down to about 40 lb. To fasten a stem to its head, a piece of blanket is placed between them to prevent a rust joint forming. The shank of a shoe, if it is to work wet, has strips of soft wood bound round it. The stem is then raised slightly and dropped to drive the shank into the socket. This is repeated with greater lifts till the shoe is firmly fixed.

To hang up a stamp when cleaning-up or making repairs, a strong tapering piece of wood faced with iron is placed on the cam so as to raise the stamp higher than usual, when it is immediately hung up either by a hooked iron which rests on two horizontal bars, one on each side of the stem; or better still, as the hook being loose is liable to fall, a wooden finger, fixed in a socket which swings on a jack-shaft at the back of the battery is pushed under the tappet. Stamps can be released by reversing the process. The capacity of a stamp varies with its weight, the class of ore, the size of the feed, and size of the discharge. Stamps may be driven by belt or gearing. A belt is better and is made to grip by a tightening pulley.

Grinding

Grinding machines have a rubbing motion which tears the particles apart. Used for reducing fine ore ½ to ¼ in. to sand and pulp.

The Chilian mill is the parent of this type and is a development of the old arastra, the drag stones being replaced by rollers. These rollers are generally two in number, but some machines have three, or four. As the rolls or runners have to revolve in a circular path, the outer edge has to travel over more ground than the inner edge, consequently the outer part slips, while the inner tends to drag, the centre part only rolling. In the Schranz mill the rolls and bottom of the pan are made conical so as to avoid the slip. The rollers are attached to a spindle which generally revolves with the rollers, or the spindle may remain stationary and the pan revolve, the rollers moving by friction.

Crushing may be carried out wet or dry. The pulp is discharged through screens at the side of the pan. The gearing may be above or below the pan. It is easier to get at when above, and is sometimes more convenient for driving purposes, but if grinding gold ore, the gearing should be below to avoid any chance of grease from getting mixed up with the ore which would sicken mercury. Chilian mills run 6 to 33 times per minute, they weigh from 25 to 30 tons, and require 25 to 30 horse-power. For slow running mills the rollers are sometimes placed at different distances from the centre so as to distribute the work over a wider die, but this would not do for fast running machines as it would unbalance them. The best grinding efficiency is obtained with a bed of 3 in. of feed. The ploughs which direct the feed under the rollers should be set so that they are not more than one inch above the top of the die. Chilian mills have been largely displaced in the bigger ore-dressing plants, but they are much used for grinding and mixing clay for brickmaking and earthenware goods, as they distribute moisture better through the clay.

Grinding Pans. These machines are designed to grind from $\frac{1}{20}$ to $\frac{1}{60}$ in. and are generally used in connection with the amalgamation of gold. The pans are cast iron, generally 5 ft in diameter though sometimes up to 8 ft—if too large they are difficult to keep level—and 2½ to 3 ft deep. The centre of the pan consists of a hollow cone through which passes a vertical shaft driven by bevel gearing from below. The muller-plate is a horizontal disc attached to the bottom of a hollow cone or yoke which fits over the cone of the pan itself. It rests on the vertical shaft which transmits motion to the muller, and its height can be regulated by a hand-screw. Pieces are cut out of the side of the yoke both to lighten it and to permit communication between the inside and outside of the yoke. To the under-side of the muller-plate are attached several shoes forming sectors of a disc. Dies of a similar shape are attached to the bottom of the pan. The shoes and dies, which vary in number, are two to three inches thick when new. They must not be so heavy as to be awkward to handle in a confined space. Several curved wrought-iron wings or guides are keyed on the inside of the pan above the muller, their object being to deflect the pulp inward and downward as it rises towards the rim of the pan by centrifugal force. The pulp passes through the holes in the yoke and under the muller till it is thoroughly reduced. Pans may be worked in charges or continuously. In the latter case the pan is provided with a small classifier. The proportion of solids to water in the pulp should be 25 to 33 for efficient grinding. If the pulp is too thin, it moves too rapidly. Grinding generally takes 1 to 1½ hours. A 5 ft pan runs at 50 to 62 r.p.m. and requires 4½ h.p. An 8 ft pan is run more slowly, 27 r.p.m., and requires 6 to 11 h.p. Coarse sand grinds best at high speed, and fine sand at medium speed.

Disc Crushers. These are used for intermediate crushing instead of rolls. There may be two or three discs to a machine, and the discs may be arranged vertically or horizontally. One or both discs may revolve. One or both discs are saucer-shaped. The crushing may be wet or dry. The chief merit of a disc crusher as compared with rolls lies in the fact that for the whole of their life the discs remain practically smooth without any tendency to form corrugations, and the uniformity of the product is maintained throughout. Their cost for upkeep is very low and their output will compare favourably with any other secondary breaker.

Berdan Pan. This is a pan, generally 5 ft diam., mounted on a shaft set at an angle which revolves 10 to 15 times per minute. The material may be fed in charges or continuously. The feed should pass through a ¼ in. mesh. The amount of water passing through the pan regulates the fineness to which the material can be ground. A large iron ball weighing about 120 lb rolls about in the pan as it revolves. This is sometimes replaced by a drag the bottom of which is cast to conform to the shape of the pan: but is not to be recommended. The Berdan pan is mostly used when cleaning gold amalgam, or in small mills for regrinding concentrates from tables.

Impact

Ball Mills. Types: Krupp, Marcy, and Ferraris.

These are a drum class in which balls, generally of iron, are lifted up and cascaded down on ore. An advantage of this class of mill is that it combines in one machine the means of pulverizing and screening the ore, at the same time returning the oversize. It occupies comparatively little room either superficially or in height. It will receive comparatively large pieces and reduces it to a fine state in one operation. It consumes relatively little power per ton of ore. Its chief disadvantage is the large consumption of steel through wear of balls and lining which amounts to 0·5 to 0·7 lb of steel per ton of ore crushed. The weight of the balls depends on the size of the machine and the horse-power available. The best size of ore to feed into a ball mill is 2½ in., as then the ore assumes a globular form and wears away gradually to smaller balls. If the feed is too fine, the balls falling into it become bedded without doing effective work, so when fine material is being crushed the mill has to be run low in ore. If worked too slowly, the balls run over each other; if run too fast the balls cling to the rim. When run at a suitable speed, the balls near the periphery do not travel so fast as the drum, especially when working dry, and the mill gives a characteristic dull roar. Ball mills are given about 20 r.p.m. They make a granular, not a slimy product. The loading of ball mills is about 45 per cent of the mill volume. The balls may be of forged high carbon-steel, manganese-steel, chrome-steel, or flint pebbles. Four to 6 in. for coarse feed, and 2 to 4 in. for fine. The liners are also made of similar steels and may be smooth, step, double step, wavy, rib, or ship-lap types. The power required to start rotary grinding mills is equal to 1½ times the full load torque. If left to settle the balls and feed may get cemented together so that it takes two or three revolutions before they get loose. Ores are generally ground wet, but non-metallic minerals are usually ground dry. They are used for grinding cement, clay, and soft ores.

Hardinge Conical Mill. This consists of two cones connected longitudinally by a cylinder. The cone at the feed end is shorter and has steeper sides than that of the exit end. It rotates at the rate of 26 r.p.m. and consumes about 40 h.p. It responds admirably to coarse feeding up to a point where from 12 to 15 per cent of the discharge is retained on the screen: after that stage is passed choking takes place, the cascading pebbles are retarded by the slush which binds the pebbles, and finally the discharge includes pebbles of all sizes. When the ore is coarse little or no water is required. Excess of water breaks the fall of the balls and washes away fine particles. A water pipe inserted through the discharge trunnion for 18 to 24 in. fitted with ¾ in. nozzles so directed that a spray impinges on the rising side of the pulp allows a proper classification and discharge to take place. The lining may be of flint blocks or a combination of flint ribbed with steel plates. The size of the mill is given by the greater diameter of the cone in feet. A pitch of 5 to 7° is found to be sufficient to cause balls to segregate according to size. The capacity and power required depend on the weight of the pebble charge, which will vary with the hardness of the mineral to be ground, the size of the feed and of the finished product. This mill is suitable for pulverizing coal to be bürnt in boilers or reverberatory furnaces, also for cement. Both coal and cement have to be ground finer than most ores so that 85 to 95 per cent will pass through 200 mesh.

Tube Mills. Types: Allis-Chalmers, Davidsen, and Abbe. These machines are closely allied to ball mills inasmuch as they are cylindrical in shape and the feed is crushed by balls. They are principally used for grinding-jig middlings, sand from gravity stamps, or material for flotation. When reducing after gravity stamps, the screens of the mortar box may be larger, say 8 mesh, which increases the capacity of the stamps, as the tube mill is more suited for fine grinding. Tube mills are set at a slight angle. One tube mill 5 ft diam. by 22 ft long will cope with the output of 20 stamps. The longer the tube, the finer the product. The feed is generally crushed wet, but can be crushed dry, in which case more horse-power is required. The speed of the cylinder varies from 24 to 39 r.p.m. depending on the size of the mill. The pebbles used should be of a size suitable for the feed: 5 in. pebbles may be used for very coarse material, ¼ in.: but pebbles 2 to 3 in. are suitable for sand from a battery. Present practice is to fill from 50 to 60 per cent of the interior volume of the mill with pebbles. The pulp should be thick enough to coat each pebble with a paste, but should not be so thick as to impede unduly the movement of the pebbles among themselves. The moisture recommended is about 39 per cent. Flint pebbles are preferred to steel balls as being lighter, more can be used at a time and so are better distributed. The pebbles must be hard and tough. The flints obtained from France or Denmark (so-called Greenland flints) are used, but where these are too expensive certain igneous rocks such as diorite, andesite, gabbro, also rhodonite, have been used. The toughness coefficient should not be below 18.

The best operating conditions depend on the diameter of the mill, the ratio of diameter to length, the number of revolutions per minute, the type and size of grinding media, the volume of charge, the circulating load, the pulp level, the pulp consistency, the liner design and the size of feed and product. A pebble lode of 4 to 6 in. above the centre line of the tube mill, and 4 in. below the centre line for ball mills, seem to be best. If there are too many large balls in a given mill, power is wasted; if the balls are too small, inefficient grinding results. The largest ball should be such that it can crush the largest particle at one blow. The pebbles often assume queer shapes, probably brought about by the smaller balls ceasing to revolve, thus getting ground.

Rod Mills. Types: Marcy, Allis-Chalmers, Cole-Bergman.

The rod mill consists of a cylinder like that of a ball mill, but rods of high carbon steel are used instead of balls. The rods are 3 to 4 in. in diameter. Small or bent rods are not efficient, while broken rods are a positive detriment. Small rods should be removed before they break. In rod mills the grinding takes place along a line instead of at a point as in the case of balls. Rods give more of a rolling and less of a cascading action than balls and cannot compete with balls for grinding hard and tough material. Rod mills work best with thick pulp, viscous enough to coat the rods to some extent. The larger rods give greater capacity and efficiency. The power consumption of a rod mill is about 75 per cent, that of a ball mill of the same capacity. For feed of 1 in., on which a rod mill will do its best work, the initial charge of rods is 50 per cent of 4 in., 25 per cent 3 in., and 25 per cent 2½ in. A 5 by 10 ft Allis-Chalmers rod mill is estimated to have a capacity of approximately 200 tons when grinding from ⅜ to 60 mesh. Balls cost about double that of rods. The waved type of liner gives a better result than a smooth surface. Rod mills are given about 20 r.p.m.

Carr's Disintegrator is an impact machine used for reducing dry soft materials, such as coal required for coke making, or for the manufacture of briquettes. There should not be more than 4 to 6 per cent moisture present or else the machine requires cleaning too often, which takes about an hour and a half. The machine consists of two wrought-iron rings mounted vertically on separate shafts but which continue in the same straight line. One shaft revolves 450 to 600 times per minute, while the other revolves in the opposite direction at the rate of 300 to 400 per min. Each ring has two circles of round steel beaters from 1 to 1¾ in. diam. The two circles on the same disc are 6 in. apart, and the beaters are 3 in. apart. The beaters of the two rings interlock with one another. The material to be reduced is fed through the centre. The whole machine is enclosed in a sheet-iron housing. The machines are made in sizes from 2 ft 6 in. to 6 ft 3 in. A 4 ft machine will reduce 15 tons of coal an hour, which should not be fed in pieces larger than ¾ in. The horse-power required varies from 8 to 16.

The Electric Ear. Experience tells one, by the character and intensity of sound, whether a grinding machine is working at its best capacity, is under-loaded, or over-loaded, whether grinding coarse or fine, wet or dry. The sound also varies with the class of material. In a ball mill, if two balls hit each other or the lining, the maximum of sound is produced and energy is wasted. If over-fed, the balls become cushioned, so again energy is wasted. Even a slight change in sound intensity makes an appreciable change in capacity.

The electric ear consists of a microphone placed under the mill where it picks up sounds, which are converted into electrical energy in a control cabinet. This energy is used to actuate the feeder, so the mill conditions are held constant at any desired noise level. In this way the running of a mill does not depend on the hearing of the mill-man who may not happen to be at hand to make the necessary alterations. Tests are made to ascertain the sound which represents the best result, and the electric ear is set accordingly.

Sizing and Classifying

Sizing is necessary in order to prepare ore for subsequent treatment. It also saves unnecessary wear and tear of the crushing machinery by eliminating fines, and this also means that less crushing machinery is necessary. The class of screen required depends on the nature of the material to be treated, whether flaky, granular, acicular, clayey, sticky, dry or wet; it also depends on the permissible oversize and undersize, the average size of feed, and whether the feed is intermittent or continuous.

Classifying is used for material too fine to be separated by sizing. Ore is much easier to treat mechanically when of uniform size, as then the ore makes an open bed through which water penetrates evenly. Particles of equal size, but of different sp. gr., tend to arrange themselves according to their sp. gr., but if of varying size also according to the law of equal falling particles. This law is, that bodies falling in a fluid, fall at a speed proportional to the weight divided by the resistance.

Now the weight of a body is proportional to the volume and increases much more rapidly than the resistance which is made up of three separate forces, viz. the frictional resistance of the fluid, which is proportional to the total lateral surface exposed, the vertical reaction (floating force) of the fluid, which is equal to the weight of the fluid displaced and the cohesive force of the water, which must be overcome in order that the body shall sink. Consequently, concentrates from unsized material will not be so clean as when sized. In most apparatus the fall is too short to allow of complete separation of equal falling classes, and the fall is further retarded by the friction of the particles upon one another, known as "hindered falling."

In America, anthracite coal is separated into the following sizes: large coal, lump (seldom used), broken, egg, stove, chestnut (passes over 1 in. screen); small coal: pea $\frac{5}{8}$ in., buckwheat $\frac{1}{2}$ in., rice $\frac{3}{8}$ in., barley $\frac{1}{4}$ in., culm or slack.

Bituminous coal may be divided into large, lump, round or screened coal (that which passes over $\frac{3}{4}$ in. screens); that which passes between the bars is known as slack, small or fine coal. Nuts is a name given to coal that passes between bars spaced $\frac{3}{4}$ in. apart but over $\frac{3}{16}$ mesh. It is made chiefly for mechanical stokers. "Duff" is what coal is left after that above nut size has been separated, and is largely used for converting into coke.

Screens may be stationary, rotary or shaking, and may be arranged horizontally or at an angle.

Grizzlies are used for separating the oversize from the undersize of run-of-mine coal or ore. They are stationary screens made of iron bars placed lengthways at an angle not less than 40° but if placed at too steep an angle, it may be necessary to have finger weights at the bottom to retard the fall. The bars are trapesoidal in cross-section, the narrower portion being below to prevent the ore wedging. The bars may be threaded on rods and kept apart by short pieces of tubing, or better still each bar rests in notches cut out of cross-bars. Grizzlies are 3 to 6 ft wide, and 8 to 15 ft long. The space between the bars is generally 1½ to 2 in. Sometimes alternate bars are made to move up and down by an eccentric which helps to keep the bars from getting clogged.

Riddles. These are portable frames with iron rods or coarse mesh, which are propped up at an angle and against which material is thrown.

Trommels or Drums. This type of screen may be cylindrical, polygonal, or conical. The frame is covered with punched iron or wire-cloth according to the size required. It generally revolves on an axle, but if heavy may revolve on trunnions. The axle of cylindrical and polygonal drums is inclined at an angle of ½ to 1 in. in twelve, are from 2 ft to 5 ft in diameter, and 5 to 10 ft long. The number of revolutions varies from 8 to 30 per minute but is generally 16 to 20. The axle of conical drums is horizontal, their shape allowing the oversize to escape at the greater diameter. One may have screens of different sizes on the same drum, but it is better to have separate drums for each size. One objection to having more than one size on a drum is that most of the wear falls on the first screen which is the finest. Sometimes drums are placed concentrically one inside the other to save space, when the coarser screen is placed in the centre and receives the feed.

Screening is better carried out wet than dry, when water is not detrimental to subsequent treatment, since water lessens the wear and tear, and keeps down the dust, doing away with the necessity of housing. Jets of water are played on the drum from either the inside or outside, or both. The sparge pipe has several holes in it. If there are too many, those which are unnecessary are plugged with wooden pegs. The rotary motion of the drum stirs the material so thoroughly that the fine particles are easily separated from the coarse. Trommels require but little head room, and comparatively low power to drive, 1 to 2 h.p. The maintenance cost is low. The depth of feed should be kept at about 1 in. The feed end of the trommel is partly closed to prevent the material from falling out.

In horizontal screens the holes are made the size required for the screened product. In the case of cylindrical screens the apertures are made one-third larger than required, to allow for the diminished area horizontally. Screens inclined at an angle of 20 to 40° have holes double that required for horizontal screens to obtain the same results. Revolving screens only have one-third their area in effective operation at one time. Inclined screens, while using the whole width of the screen at the top, tend to centralize lower down. The efficiency of a screen is always decreased with heavy feeding as the fine material has difficulty in finding its way through to the screen.

Shaking and Vibrating Screens. These are not much used in ore-dressing plants. They are more complicated than drums, take up more floor space, and add to the jarring of the building. But against these disadvantages their capacity for equal areas is greater than for revolving screens; they can be placed at a flatter angle than a stationary screen, and therefore require less height. There are many varieties of vibrating screens. They may be horizontal or inclined, and have from one to three decks. They are used for fine material which would pack on machines which had a simple reciprocating motion. The screen is made of wire-cloth. The feed should cover the whole surface uniformly, otherwise a space is left bare. The feed should be regular, and each machine should have its own motor.

Screens sometimes form a part of a machine, as in the case of stamp batteries, Huntingdon mill, and Chilian mill. When this is so, the screen consists of punched iron or woven-wire cloth. Punched metal is best suited for coarse material, and wire-cloth for finer material. When ordering wire-cloth, one should not only mention the number of holes to the linear inch, but also the gauge of the wire. Unfortunately the English and American standard screens do not agree in the dimensions of wires and size of apertures.

Particles which pass through a hole 0·01 in. diameter are too small to separate economically by sizing. Very fine screens are not suitable for work about ore-dressing plants: they are too delicate and become expensive on account of frequent breakages, so hydraulic classifiers are used to prepare fine material for further concentration, where the particles are not of the same size, but of the same falling class. There are various forms of classifiers, the chief types are spitzkasten, spitzluten, and cone separators.

The Spitzkasten or Pointed Box is pyramidal in shape. They are arranged in series, each succeeding box being larger than that preceding it, the increased area slowing down the current and allowing finer particles to settle. A baffle board is placed near the feed end to prevent surface currents. A rising current of water from the bottom prevents the slime from settling. The concentrates which settle against the upward current fall into a T-pipe below and are washed out.

Spitzluten. These are also arranged in series. A unit consists of two V-shaped boxes, one of which fits inside the other. A space capable of adjustment is left between the two which varies with the nature of the work to be done. The main current of the pulp flows down one side and up the other after depositing the particles which are capable of sinking against the rising current. The bottom below the spitzlute proper is pyramidal in shape so as to facilitate the discharge of the concentrates. An upward current of water is used as in the spitzkasten.

Cone Classifiers. These are a modification of the spitzlute which they have largely displaced. The apparatus consists of a cast-iron cone and an inner boiler plate cone or cylinder open at the bottom. In some forms the inner cone can be adjusted like the inner box of the spitzlute. The pulp flows down through the opening of the inner cone to the space left between the two cones. Here the pulp meets with a rising current of water against which the particles have to settle if sufficiently heavy, and they are drawn off through pipes leading to their particular concentrator, while the lighter material passes along a launder to the next larger cone. There are many modifications of the cone classifier. F. D. Stephen leads the pulp down a pipe inside the inner cone. At the bottom of the pipe is a flange to which a baffle-plate is attached by bolts leaving a space between them. The baffle-plate acts as a deflector and directs the pulp upward. W. G. Mosher lets the pulp flow directly into the cone and allows the fluid to rise up through the central cylinder from which it overflows, thus reversing the usual course. The Allen classifier is designed to work automatically.

Screen Sizes. As a means of finding the perforations required in a revolving screen, and the sizes they will produce, the following table will be useful.

Ring Size	Cube Size	Screen Holes	Ring Size	Cube Size	Screen Holes
3/16	1/8	1/4	1 3/4	1 1/4	2
1/4	3/16	3/8	2	1 7/16	2 1/4
3/8	1/4	1/2	2 1/4	1 3/4	3
1/2	3/8	5/8	3	2 1/8	3 3/4
5/8	7/16	3/4	3 1/2	2 1/2	4
3/4	1/2	7/8	4	2 13/16	5
7/8	5/8	1	4 1/2	3 3/16	5 1/2
1	11/16	1 1/4	5	3 1/2	6
1 1/4	7/8	1 1/2	5 1/2	3 15/16	6 3/4
1 1/2	1 1/16	1 3/4	6	4 1/4	7 1/2

Concentration

There are two media in which concentration can take place: water and air. The former is preferred, but if not available air is used to take its place.

The material to be concentrated should be graded. It may be coarse, sand, or slime; and the class of machine varies accordingly.

Coarse material is best treated in some form of jig; the fine on some kind of table.

There are two main types of jigs. (1) Those where a tray with a perforated bottom is moved up and down in a box of water, as with hand jigs and the Handcock jig. (2) The piston jig of the Hartz type. They both have a pulsating motion.

Hand Jig. This may be used for concentrating coarse material up to 3 in. The jig-box must be perfectly level. The frame to which it is attached has a lever or jig-pole 6 or 7 ft long worked by men standing on a spring board who give it 90 strokes per min of 2 to 3 in. long. The perforations in the screen are 0·4 in. in diameter. The fines pass into the hutch and can be drawn out through a 2 in. spigot. After about 200 strokes, the waste on the top is scraped off and thrown away; and if there are sufficient concentrates, they are removed; otherwise a fresh charge is fed on the top of them. Two men on a hand-jig can treat 0·6 to 0·7 tons per hour. When hand-jigs are used for fine material, the screen is 8 mesh of No. 16 gauge steel wire. One man can operate it, the stroke being 1 in. and 112 strokes are given per min. The jig-box is kept entirely under water, so there is very little splash.

Handcock Jig. This is suitable for low-grade ores where cheap handling and large tonnage is the objective. The jig-box, which is about 20 ft by 2½ ft inside dimensions, is worked up and down by machinery. The motion is such that the material works towards one end of the machine. The amount of motion as also the inclination is controlled by hand-wheels. One jig requires about 2 h.p. and can treat as much as 175 tons daily. The concentrates pass through the screen into the hutch, while the tailings pass over the end. The capacity of a jig is in proportion to the screen area. Jigs with movable screens have a capacity of 4–9 tons/sq ft screen area.

Piston Jigs. These can be used for treating either coarse or fine material, but are modified accordingly. When used for coarse the tailings and concentrates pass out above the screen, but at different levels. For fine material the concentrates pass through the screen into the hutch. The piston jig is divided into two compartments: the divisional board projects a little below the level of the screen. The piston occupies one side and the jig-box the other. Pulsations are given to the water which agitates the material in the jig-box causing the lighter material to pass up higher than the heavier with the downward stroke, and the heavier to sink lower than the lighter with the upward stroke of the piston. A jig may have from one to four cells depending on the classes of minerals to be separated. Besides concentrates of different minerals and waste, there may be middlings in coarse jigging which require further reduction to free particles for concentration on some other machine. When there is more than one cell, there is a vertical division between them so that the motion imparted to the water of one division does not interfere with that of its neighbour.

The pistons do not all move up and down at the same time. For coarse jigging the holes in the screen are smaller than the material to be treated. For fine jigging the holes are larger so that concentrates can pass through. The screens are supported on iron or wooden grids. The screen of the first cell is a little higher than that of the next, and so on for each cell. The lighter portion of each jig-box passes over a low weir into the next compartment, while the heavier material either passes through an adjustable slit below the level of the lighter material, or there may be a pipe passing through the bottom of the screen to convey the heavier material away. This pipe is protected from the lighter material from entering it by a shield, the bottom of which reaches a little below the normal height of the concentrates.

In the case of fine jigging a bed is made 1 to 2 in. thick of comparatively large pieces of similar mineral to that being jigged. The ratio of the size of the bedding to the size of the feed is generally 3 to 1. The ragging may be made out of $\frac{5}{16}$ in. round bar-iron cut into lengths of $\frac{3}{8}$ in., or may be shot. The ragging acts as valves allowing the fine material to pass between the pieces and through the mesh of the screen. The pistons, which move vertically, fit loosely and sometimes have auger holes through them in order to reduce the suction of the back stroke. In the ordinary Hartz jig the up and down strokes occupy the same length of time by means of double eccentrics, the inner one being about $\frac{1}{2}$ in. out of centre with the shafting, and the other eccentric about $\frac{1}{2}$ in. out of centre with the inner eccentric. With high-speed jigs, the suction of the back stroke is considerable. To overcome this an elbow-like arrangement has been devised whereby the down stroke is made to occupy only one-third of the time of one revolution of the countershaft. Another arrangement with the same object in view is the slide. A slotted lever is keyed to the rocker shaft: in the slot is a freely moving block. As the shaft revolves, the block slides backward and forward in the slot.

If water is scarce, the tailings may be discharged by a revolving shovel-wheel, or oscillating scoop.

The number of strokes for a coarse jig is about 75 per min, and for fine jigs 200 or even 300 for very fine material. Coarse jigs may have a stroke of $5\frac{1}{4}$ in., but the usual maximum is 2 in. The ability to make tailings is the criterion as to whether or not a roughing operation is possible.

If a fine jig is not working properly, see if the hutch has become filled by the bursting of the screen. If that is not the case, try scratching the screen. If that is not sufficient, shovel all the material off the screen and beat it with an old piece of belting to free the mesh from any particles which may be blinding it. The trouble may be due to overfeeding or the want of more water under the screen. The jig-box may be made of iron or wood. Wood lasts longer than iron. The bottom should be rounded the better to guide water under the screen. This is easily done when jigs are made of iron, but not so easy with wood, so the longer sides are generally made to meet V-shaped as the next best thing.

The capacity of jigs with fixed screens is 0·5 to 2 tons per 24 hours per sq ft screen area.

Jigs are sometimes used on dredges in preference to riffles as they do not pack up with black sand.

Jigs are also used for washing coal; but in this case, the coal being lighter than shale passes over the top, while the shale sinks to the bottom. What is known as a "fluid mass" is water made apparently dense by the addition of fine sand kept in constant agitation which is used to float off coal. The dimensions of a coal jig are 12 ft long and 6 ft wide. It has two cells. The cell into which the coal is fed is called the "rougher," the other is called the "finisher": both cells are the same size. The rougher cell is 12 in. higher than the finisher. The length of stroke of rougher and finisher is 6 and 5 in. respectively. An apron at the discharge end of the finisher conducts the coal and water over a 4 by 5 ft grizzly.

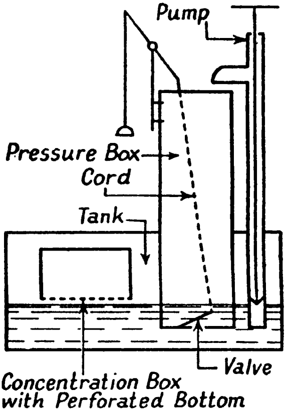


FIG. 74. THE WILLOUGHBY

The Willoughby. This is used for cleaning alluvial tin collected in sluice boxes, which is not rich enough to send direct to smelters. The machine consists of a tank or reservoir 5 or 6 ft long, 2½ ft wide, and 2½ ft deep, inside of which is a duplex hand pump; a pressure box or water tower 5 ft high, and 30 in. square section; and a concentrating box 2 ft square and 18 in. deep, the bottom of which is punched metal, preferably copper or some metal that will not rust. The perforations must be sufficiently small (about $\frac{1}{32}$ in.) to retain the feed. The bottom of the pressure box as well as the concentrating box, is raised above the bottom of the reservoir. There is a valve at the bottom of the pressure box which can be opened by pulling a cord. The ore to be treated is placed in the concentrating box to a depth of about 6 in. (i.e. about 4 cwt). Water is pumped into the pressure box, the valve opened, and water under pressure passes through the charge which agitates it, giving the heavier portion a chance to sink and the lighter material to rise on the top. Some of the water overflows, and some drains through the perforations when the water in the reservoir is again pumped into the pressure box. This procedure is repeated about five times: the material is then removed in three grades. The lighter impurities (black and white sands) are scraped off and thrown away. The next layer consists of mixed tin and impurities, which is treated in a streaming-down box. The lowest layer is clean tin ore which averages 76 per cent metallic tin. This is dried on a hot iron plate and bagged. The machine is worked by one man, and its capacity is about 2 tons in 10 hours.

The Keeve or Tossing Tub. This is employed for enriching fine tin concentrates from tables. It consists of a wooden tub made of 2 in. staves which is 4 ft in diameter at the top. Passing through the tub is a 3 in. spindle, on the top of which is a yoke carrying eight flat iron stirrers 2 in. wide. A charge consists of ¾ ton. During the process of filling the tub, the stirrers rotate 48 times per minute. They are then withdrawn and two heavy hammers commence to work against the side of the tub, delivering from 80 to 150 blows per minute while settlement takes place. At the end of 40 or 50 minutes, the water is siphoned off, and the contents of the tub classified into tops, middles, and bottoms. The concentrates vary from 40 to 70 per cent tin.

When air is employed as a medium for concentration it may be used as a continuous blast which grades particles subjected to it, or as intermittent pulsations.

The continuous blast is used for separating coal dust from shale. The material may be blown through chambers of increasing cubic contents in which dust of different settling capacities is deposited, or else the material travelling on a belt is subjected to blasts of different velocities. If the mixture is blown up an inclined plane, the floor of which is made of overlapping boards like a venetian blind, the coal will be kept in suspension while the shale will sooner or later present its edge to the blast, fall down, and escape between the boards.

The pulsation blast is used in some form of pneumatic jig or dry blower, largely used to concentrate alluvial gold, tinstone, wolfram, precious stones and other substances in places where water is not available. Dry blowers on alluvial fields are generally mounted on wheelbarrows so that they can be readily moved from one place to another. They consist of an inclined coarse screen on to which the alluvial is fed so as to separate the pebbles. This is caused to shake by means of a cam or rod connected to the handle that works the bellows.

The fine material that passes through the coarse screen falls on an inclined perforated plate forming the top of the wind chest of the bellows. Riffles are placed horizontally across this to retain the heavier particles, the blasts of air separating the lighter particles. The capacity of such a machine under ordinary conditions is about four or five loads a day.

Magnetic Concentration

When a mineral is attracted by a magnet, it is said to be paramagnetic. A mineral not attracted by a magnet is said to be diamagnetic. The paramagnetic minerals are few in number and vary considerably in their power of attraction. The minerals mostly attracted by a magnet are those containing iron, e.g. magnetite, pyrrhotite, wolfram, and the dark variety of zincblende.

Magnetic Separators

Guard Magnets are used for picking up tramp iron in mills, such as hammer-heads, gads, etc., which if allowed to pass might damage some of the machinery. There are two classes: those which are automatically cleaned, and those which have to be shut down to be cleaned. The size of the magnet depends on the size and quantity of the material and tramp iron, and the rate of feed.

The automatic type consists of a drum which may be the head pulley of a conveyer belt, or a separate drum inside of which is an electro-magnet. The ore is carried away on the belt, while any iron present clings to the belt or drum till it passes the influence of the magnetic field, when it automatically falls off. The speed of the drum should be under 40 r.p.m. The feed must not be so thick that the magnet cannot attract the iron through it.

Suspension magnets, which are generally circular in shape, are placed above a conveyer belt to pick up tramp iron. The belt may travel up to 500 ft/min, though this speed is seldom reached. Not being self-cleaning, these magnets are mounted on a trolley which runs on a cross girder so that it can be removed to one side when, by switching off the electric supply, the iron drops off.

All magnetic separators require direct current, as an alternating current, being fluctuating, is not suitable.

Some minerals not magnetic in themselves, such as pyrite, hematite, limonite, and siderite can be made magnetic by roasting. Magnetic separators are made for both dry and wet feed; but mostly dry. There are many kinds of magnetic separators, the difference being mostly of a mechanical nature. One type has a drum of some non-magnetic material, such as brass, bronze or rubber, inside which is a segment (½ to ⅔) of electro-magnets. Ore is fed on the top of this. The non-magnetic particles fall off into one bin, while the magnetic particles, after passing the influence of the electro-magnets, are discharged into another. The drums are 2 to 3 ft in diameter, with a face 16 to 24 in. wide. Another type consists of a short rubber belt which revolves round pulleys at a rate of 100 to 400 ft/min. The head pulley is of steel and has a battery of electro-magnets all round the inside of it mounted on the pulley-shaft and revolving with it. The material is fed on to the belt. The non-magnetic particles fall over the end, but the magnetic particles cling to the belt till past the influence of the magnets, when they fall off into a separate bin. The magnets are wound so that adjacent poles are of opposite polarity. This has the effect of causing each magnetic particle to change its position as it is carried forward; and in this way most of the entangled non-magnetic particles are eliminated. For the best work with magnetic separators the ore must be finely crushed.

Flotation

General. The flotation process depends on surface tension of gases, liquids, and solids. The five basic functions of the process are: frothing, collecting, depressing, activating, and conditioning. A very small quantity of each reagent is required to carry out its work. Some reagents combine different functions. For instance lime is a *p*H regulator, and also a depressor, especially for pyrite. The reagents are added at different stages of the process. Each problem should be carefully tested on a fair sample of the ore to be treated, to determine the best reagents to be used and their approximate quantity before putting it into practice on a large scale.

When a mineral to be saved is finely disseminated through rock, it must be freed from the surrounding minerals by grinding. This is generally done in a ball mill. The ore must be fine but not slimy for flotation purposes—between 40 and 100 mesh. The pulp density for mixed sand and slime is 30 per cent solids, and for slimes alone 15 per cent solids.

The pulp is agitated mechanically together with floating material and air. The speed of the impeller is about 350 to 400 r.p.m. The froth overflows at the sides, and the tailings are drawn off through a spigot at the bottom.

Frothers. Pure water will not froth; there must be some frothing agent present which is at least partly soluble in water. The froth must have sufficient buoyancy to carry the mineral particles to the surface, and stable enough to last until it passes the lip of the flotation machine. Frothers in use are all of an organic nature.

Pine oil is one of the most common frothing reagents. It is used for all types of metallic and some non-metallic minerals It produces a froth of medium body, but has practically no collecting value for metallic minerals though it has a marked collecting value for coal, graphite, molybdenite, talc, and sulphur. It is added to the flotation current in quantities varying from 0·05 to 0·20 lb/t of dry ore.

Cresylic acid produces a slightly more brittle froth than pine oil, or pine tar oils, so it requires a slightly higher concentration for the same value of froth. It has distinct collecting properties for lead and copper sulphides.

Oleic acid (red oil) and fish oil possess both frothing and collecting properties in the flotation of non-metallic minerals, e.g. calcite, apatite, fluorite, barytes, also strontianite, witherite, pyrolusite, wolframite, scheelite, and several other minerals.

Coal tar and coal tar creosote oils formerly used as primary frothers and collectors, are now used as supplementary promotors and frothers, or as froth modifiers, to stiffen froth produced by pine oil or cresylic acid. It may be necessary to regulate the froth, if deficient, by using some frothing agent, e.g. cresylic acid; or if there is an excess of froth, lower it to the required amount by adding a non-frothing collector, e.g. a xanthate.

Eucalyptus oil is also used.

Collectors. Collection is carried out by the attachment of certain minerals to bubbles of a collecting agent. Collectors collect several minerals, but do not select them from one another. They formerly consisted of wood- and coal-tar creosotes and petroleum products, but these have mostly given place to synthetic reagents which give much greater efficiency. Fatty acids also serve as efficient collectors for certain oxidized ores. The most common collector for oxidized and non-metallic ores is oleic acid. It has been used with good results on oxidized copper and lead ores, calcite and a number of non-sulphide minerals. It makes no difference whether this or any other organic acid is free or its soap. Palmitic acid is also employed for this purpose. Most of the fatty acids and their soaps have marked frothing properties, but are usually looked on as collectors and are used in conjunction with pine oil.

For collecting sulphides, the chief collectors are the xanthates and dithiophosphates or "aerofloat." For most purposes ethyl xanthate is satisfactory; the butyl and amyl xanthates are more powerful, but their use is only justified for certain ores on account of their high cost. It is added to the grinding or conditioner circuits from 0·05 to 0·15 lb/t of dry ore.

Modifiers, broadly speaking, include all reagents other than collectors and frothers which, when added to the pulp, cause a mineral to float more readily than in the absence of the reagent. Lauryl amine hydrochloride (L Am HCl) when used in small amounts is a good collector for the selection flotation of galena and sphalerite from the common gangue minerals. Cresylic acid has a distinct collecting property for lead and copper sulphides.

Depressants. Depressants distinguish between different minerals. By the addition of depressors fine particles of gangue are kept in suspension which would otherwise adhere to sulphide grains and obstruct the action of the collector. Depressants are employed in selective flotation to inhibit the flotation of certain minerals. In a complex ore of galena, sphalerite, and pyrite, though the sphalerite is not nearly so easy to float as galena, yet some of it will contaminate the galena float unless a depressant is used. The standard depressant for zinc and iron sulphides is a cyanide (sodium, potassium, or calcium) sometimes combined with zinc sulphate in an alkaline circuit. The sodium cyanide seldom exceeds 0·10–0·25 lb. The zinc sulphate is about three times that of the cyanide. Another, but less usual method, is to float a bulk concentrate and then depress the galena with sodium or potassium bichromate; 1 to 5 lb/t, and float the sphalerite and copper. Pyrite and most sulphide minerals, with the exception of galena and chalcopyrite, are easily floated with most collectors, and consequently contaminate copper, lead, and gold concentrates. The most powerful depressants for pyrite are lime, ferrous sulphate, and thiosulphate. Lime is fed as milk of lime and is mostly used on account of being cheap. It also acts as a pH regulator and is used in sufficient quantity to make the pulp quite alkaline. Lime also depresses gold. In addition to increasing the floatability of copper sulphides, by depressing the pyrite, it also aids in the selective flotation of copper sulphides from pyrite.

Sodium silicate (water glass) is a depressant of gangue minerals. It can be added at any part of the flotation circuit, generally in the form of a 5 to 10 per cent solution. An excess in the flotation circuit generally makes a brittle type of froth and tends to lower recovery. Sodium silicate is both a depressant and a dispersor. It inhibits the flotation of galena.

Tannin or tannic acid acts as a depressor by coating certain minerals with a protective colloidal coating. When calcite and fluorspar are associated, the fluorspar can be floated off. Starch acts in a similar manner: it depresses talc and serpentine in the flotation of gold ore.

Activators. Copper sulphate is used for activating sphalerite after it has been depressed by cyanide used in quantities between 0·05 and 0·20. In fact, sphalerite will not readily respond to flotation in the absence of this reagent. When arsenopyrite is present in an appreciable amount and it is necessary to float it off, use a combination of 0·05 to 1·0 lb copper sulphate, and soda ash, the quantity depending on the acidity of the ore, but usually does not exceed 1·3 lb/t of dry ore.

Sodium sulphide is used to form a coating of sulphide on oxidized ores of copper and lead so as to make them float. The amount of the reagent used must be carefully determined, for if in excess, it acts as a depressant. Sufficient time must also be allowed for conditioning. This is roughly proportional to the amount of reagent which is added to any one point in the circuit. To sulphidize bauxite, at least ten minutes are required. Sodium sulphide is also a depressant for free gold and sulphide silver minerals. In the case of an ore containing manganese dioxide, basic sulphate, or lead peroxide, these minerals tend to use up sodium sulphide before lead carbonate is satisfactorily sulphidized.

Conditioners. A flotation pulp may have chemical or physical characteristics which do not give the best results with given reagents. The usual procedure is to add a corrective chemical. The trouble is generally overcome by regulating the pH of the pulp. In general practice the range of pH is about 8 or 9, and as most pulps tend to be more acid than this, the use of an alkaline reagent is almost universal. Lime is the most common because it is cheap; but lime has other effects. Too little lime causes an excessive froth: too much lime may kill the froth. Soda ash (sodium carbonate) can be used in the place of lime, but as it costs more it is only used where lime would be harmful. As soda ash does not depress free gold or galena, it is used for gold ores and in lead circuits when treating complex lead-zinc ores. Sodium hydroxide can also be used, but is more expensive than soda ash and offers no advantage over it. Acid circuits are now rarely used, but when they are it is sulphuric acid which is the reagent employed from 0·05 to 0·2 lb/t of dry ore.

Desiccation

In some cases it is necessary to reduce the moisture in an ore, especially if it has to be transported, for the freight on water is the same as if it were ore. Moreover, moisture may set up chemical action as in the case of pyrites, which if bagged may rot the material of which the bag is made, thus causing loss. Damp ore tends to clog certain machines and makes it difficult to crush. One must also consider the desired degree of dryness. If all the moisture is driven off concentrates, they become dusty and some may become lost through air currents. It is more difficult to dry ore down to ½ per cent than to 2 per cent, the latter being dry enough in most cases. There is no danger of igniting inflammable material which contains a considerable amount of water, and in drying such material, it is advisable to do so in two stages. The first can be fired heavily, the second or finishing is fired lightly. Before deciding on the method of drying, one must consider whether the mineral will be injured by a high temperature or by the products of combustion passing over or through it.

In dry climates much moisture can be eliminated by exposure to the weather.

Very wet material may get rid of much of its water on a draining belt. This is a rubber belt placed at an angle which circulates and at the same time serves as a conveyer.

Though most ores can be dried by the direct passage of the products of combustion from a furnace coming in contact with them in the case of some industries, such as certain clays and glass sands, this is not permitted on account of the danger of discoloration.

Small quantities of concentrates are sometimes dried on plates under which a fire is built or the products of combustion are allowed to pass. This is not only slow, but extremely wasteful of heat.

Drying by direct heat in rotary cylinders can be used in most cases and is the cheapest method. The fire is made outside the cylinder which becomes a flue. The cylinder varies in length but is usually about 25 ft long, 7 to 8 ft diam. and is given 1 r.p.m. It is carried on friction rollers. All iron parts should be designed to allow for expansion and contraction. The steel sheets should run the length of the cylinder and there should be no cross seams. Longitudinal lifters are arranged inside the cylinder to allow the ore to rain through the heated gases.

Flotation concentrates are fine, sticky, hold a quantity of water, and are difficult to dry. It is usual to dewater them in a Dorr thickener to about 50 per cent, and then further reduce the water in vacuum filter presses.

Flash drying is mostly used for drying pulverized coal used for heating boilers or reverberatory furnaces. The powder is sucked up by a fan through a pipe together with heated air. This method is not suitable for material coarser than 30 mesh or for abrasive material which would wear out the pipe and fan. The temperature may be as high as 380°.

Dewatering

The Dorr Thickener. This apparatus is largely used for getting rid of excess water from tailings, or thickening pulp prior to concentrating or filtering. Its capacity depends on the physical nature of the material handled. The thickener consists of a tank made of iron plate or concrete. It varies in diameter from 20 to 75 ft and is 5 to 12 ft deep. Being large it is built in the open. The bottom is flat or slightly conical. A slow moving mechanism suitably supported, generally by a truss which travels on a rail round the periphery of the tank, consists of rake arms near the bottom. The rake arms can be raised or lowered. The rakes themselves are given a suitable pitch so as to direct the settled material to the centre opening. The pulp may be discharged through a suitable plug spigot, but is better handled by a diaphragm pump. The feed passes through a screen to strain off chips, etc., it then passes into the tank under the surface of the water. The feed generally enters at the centre of the tank; the clear liquid overflows continuously over the whole periphery. The cost of driving is very low. Thickeners below 50 ft diameter average ⅛ h.p. A 100 ft diam. requires 1·25 h.p., a 200 ft diam. 2·6 h.p., and a 325 ft diam. 5·3 h.p.

The Dorrco diaphragm pump is designed to handle dilute or thickened pulps. The pump is equipped with ball valves which are easily accessible, and the diaphragms can be changed in a few minutes. There are two types manufactured, one for suction, the other for pressure.

Filters. Intermediate pressure filtration, where thick cakes of solids are gradually built up, is not as a rule so suitable for ore dressing as continuous vacuum filtration which forms comparatively thin cakes. The efficiency of thin cake filtering, washing, and drying is far greater than that of a thick cake. This is due to the fact that the filtering rate of most materials decreases approximately in proportion to the square of the thickness of the cake; hence the flow becomes rapidly slower as the thickness of the cake increases. With drum filters the work is continuous and all stages of filtration are performed at each and every moment, for while the pulp is being filtered at one point, it is being discharged at another. Before determining the quantity of solid or liquid that can be handled on a filter of a certain size, one must consider the condition of the solids (whether granular, fibrous, colloidal, or a combination of any of these); also the size of the particles, and ratio of solids to liquids

The revolving drum type is partly submerged in a tank containing the material to be filtered. It may be necessary to have an agitator below the drum to keep the solid contents in suspension. The drum is given about 47 r.p.m. A vacuum is applied to one side of a pervious fabric, such as cotton cloth, monel or brass wire gauze, supported on a grid of wood or metal. Such a filter can deal with about 160 lb/sq ft/h. The cake is either blown off or scraped off.

A disc filter is sometimes used which is practically a very narrow drum.

Conveyers

There are two classes of conveyers mostly in use at mines, viz. tray and belt conveyers.

Tray Conveyers. These consist of a series of interlocked metal trays with small wheels which travel on rails. These conveyers can be made to work at any desired angle, and some types are so constructed that the trays can be made to discharge at any desired point. They require comparatively little power and very little attention. The first cost is greater than other types of conveyers; but the cost of repairs and renewals is low.

Belt Conveyers. These are made of cotton duck impregnated with rubber and then covered with a layer of rubber to keep it from rotting. The cotton fabric gives the belt strength, approximately 350 to 400 lb/in. of width per single ply of fabric. Any continued excess strain of 26 lb/in. in width per ply causes elongation of the belt, increases the tension of the rubber, and tends to draw out the belt fasteners. The number of plies depends on the tensile strength required. There is no object in having a belt unnecessarily strong as it reduces its flexibility. The number of plies also varies with the width of the belt. The diameter of the driving pulley should be correct; it should be increased 4½ in. for each ply of the belt. Driving pulleys should be lagged with belting similar to that of the belt.

The belt may be flat with side rims, but is more usually troughed by idler pulleys. These sets of idlers should be sufficiently close together to give the belt enough support, otherwise it will sag. The distance apart depends on the load carried, and the width of the belt from 4½ to 3½ ft. The grade at which the belt can be used varies from 16° for free flowing material to 23° for powdered material such as cement and tailings. The usual speed for the belt is 300 ft/min but they can be run up to 600 ft under ordinary conditions for belts 13 to 18 in. in width.

When calculating the capacity of a belt conveyer, one must allow for irregularity in feeding. A flat belt will only carry about one-third as much as a troughed one. The diameter of the carrying pulleys is of importance. For a speed of 300 ft/min and upward, they should not be less than 6 in. diameter, and for lower speeds 5 in. The idlers should be periodically inspected to see that they are properly lubricated and not clogged with dirt. The application of grease should be little and often, as an excess collects dust. The return idlers are not troughed, and are usually placed twice the distance apart of the troughing idlers. The driving pulley should be of much larger diameter than the width of the belt. The angle of the troughing carriers should be about 15° for three troughing carriers, and 10° for two. A conveyer belt 18 in. wide requires 2 h.p. for every 100 ft of travel per min, and 6 h.p. when running at 300 ft/min. The major wear on belts is at loading points, and this is aggravated if the material is fed on the conveyer at right angles to the direction of travel. The feeding chute should always be arranged so that the material contacts the belt in the same direction that the belt is moving, and the velocity of the material should be the same as that of the belt. The falling of large lumps from a height increases the wear of the rubber considerably. When sharp heavy material is conveyed, it is generally arranged that the feed passes over a grizzly, so that the fines serve as a cushion for the larger and heavier pieces. The lip of the chute should be approximately two-thirds the width of the belt to assure that the material is fed to the centre line of the belt. There should be ample clearance between the chute-lip and the belt. When setting up a belt conveyer, the driving head and return pulleys should be fixed in strict alignment with each other, and the idlers kept in alignment with these fixed points.

Misalignment may be caused by any of the following—

(*a*) Unequal adjustment of the take-up screws.
(*b*) Misalignment of the idlers.
(*c*) Fitting belt fasteners in such a manner that they are not at right angles to the edge of the belting.
(*d*) Idlers not revolving.
(*e*) Variable loading.
(*f*) Insufficient belt contact with the idlers.

The life of a conveyer belt can be considerably lengthened by rebuilding it where worn, with suitable strips cut from discarded belts and clamped to it.

Bucket Elevators. These are used for raising fine material, wet or dry, up to 1½ in., to a maximum height of 50 ft; above that it is necessary to elevate in series. The buckets may be attached to belting or chains. Chains are better for hot material, but grit gets between the links and causes much wear.

As the belts have a direct lift, they must be made stronger than a horizontal conveyer belt, and the fabric must be close-woven for security of the bucket bolts, and to stand the rougher usage. The speed of the belt should be such that the material in the buckets can be properly and completely discharged. The belt must be kept in correct alignment, bolts tightened periodically, and worn buckets and washers replaced.

Accumulation of material on one side of the boot tends to drive the belt out of correct alignment. The belt may be vertical, or better still at an angle of 80°. Elevators should be housed to avoid splashing or dust. The housing should be at least three times the width of the elevator belt so as not to cramp those making alterations or repairs, and the sides should be capable of removal.

The boot of the elevator should be above the level of the ground to facilitate emptying it. The boot is generally 2 ft deep with a 6 in. plug for the removal of feed in case of stoppage.

The top and bottom pulleys are generally 2 ft 3 in. diameter, each having a substantial crown ⅛ to ¼ in.; in all cases they should be 3 to 4 in. wider than the elevator belt. The head pulley should have a flange, but a flange would be detrimental on the boot pulley as it would retain sand within the rim which would cause wear. There should be a take-up bearing, generally at the boot pulley, but sometimes at the head pulley.

The elevator may be driven by belt, but it is better to use gearing; the ratio of spur to pinion being 4 to 1. The buckets may be spaced or overlapped. They may be U-shaped, V-shaped, or L-shaped. The L-shape has an ideal discharge but low capacity. Deep buckets have a large capacity but poor discharge. When a bucket loses its shape, it should be replaced as it will be unable to discharge properly. Ordinary lapped and riveted sheet metal buckets are not economical for handling wet crude ore. Malleable iron pressed into shape without being lapped or riveted should be used for hot or corrosive materials, but must be made of heavy gauge if for handling coarse material. V-shaped buckets are used for continuous bucket elevators if the material discharges readily, but are no good for material that packs. The edge of the buckets should just clear the edge of the receiver. With normal feed the bucket is only half full, but it is better to reckon on a quarter for ordinary duty. Buckets may be spaced 12 to 20 in. apart. When a bucket is bolted direct to a belt, there is a certain amount of play between the two which causes considerable wear, so use leather washers to keep them apart.

To strengthen the belt a piece of balata belting is placed between the bucket and the belt, and a strip of hoop iron at the back of the belt through which the fastening bolts pass. Higher speeds can be used with wet than with dry materials. If the material is coarse, it should be fed direct into the bucket from 18 in. above the centre of the pulley and let the following buckets catch any spillage. Pulp can be scooped out of the boot.

Launders. These are used to convey solids in a current of water. The solids may be transported by rolling along the bottom or carried in suspension. The diameter of the particle which any current will transport will depend partly on its shape and sp. gr. but chiefly on the velocity of the current. In order to obtain the required velocity with the least possible slope and with the least volume of liquid, it is necessary that the wetted surface should be as smooth as possible. Wood is the most convenient material and lends itself to alterations and repairs. Launders should be made twice the width of the depth of the fluid, and the depth of the fluid should be one-fifth the inside depth of the launder. If the quantity of fluid is liable to vary, it is better to make the cross-section of the launder half a hexagon. Then the diminished quantity of water is confined to a narrow section and will have a better carrying capacity.

When the particles are too large to be carried in suspension, another factor comes into force, viz. friction of the particles among themselves and against the bottom of the launder. The shape of the particles also makes a difference, for round and flat grains will move more easily on a flat surface than a rough one. To obtain the maximum efficiency, the transported particles should move at a velocity one-half that of the transporting current. When particles are in suspension, they should travel at the same rate as the liquid. If the solids pile up on themselves, it shows the force of the current is not sufficient, and it forms a rough surface. Where sufficient fall cannot be obtained, a shaking launder is sometimes used. Very fine sand or slime is occasionally conveyed through pipes, but as pipes are closed one cannot see what is going on inside, and they are not so easy to clean.

When the under portion of a pipe is worn, it can be turned so as to give a fresh surface. Sheet-iron launders are a source of constant trouble. To protect launders from wear, a wooden shell may be lined with cement made in convenient lengths for handling. A curved bottom is not so suitable as a straight bottom where part of the solids is conveyed by rolling. If the pulp is frothy, the width of the launder should be made greater than under ordinary circumstances so as to increase the surface area. If necessary to make a bend, the grade must be increased at that point to compensate for the curvature, and the bottom of the outside of the curve must be raised.

If it is desired to divide a stream into two or more branches, the feed launder should be flaired out to the full extent of the distributing box and the flow forced to pass under a baffle-plate which should be high enough to prevent the feed from slopping over, but low enough to allow feed to pass over should there be a choke-up under the baffle.

Stacking

It is necessary to dispose of tailings, mullock, and sometimes to stack ore, coke or coal. In the case of tailings, if the mill is on a slight slope, it may be possible to form a low wall of sand to impound the tailings, which wall is added to as the heap increases in height. A pipe is left in the heap to drain off the water, and this also is added to from time to time. If the tailings have to be stacked on flat ground they are first dewatered and then made into hills 200 ft high. This may be done with the help of an endless belt conveyer, endless rope railway, or an aerial ropeway.

The endless belt is supported by a bridge at an angle of 20° and is made up of a number of short sections bolted together which are added to as the heap grows. The conveyer is extended in the form of a cantilever held by guy-ropes from the upright trestles in the dump. A belt to handle 40 or 50 t/h would require a 24 in. 6 ply rubber belt. When an inclined conveyer has been carried to its economical distance, the tailings at its end are made into a bed and planks laid down for a boom stacker. This is an iron pole which rests on a steel ball in a cup-shaped receiving device which enables it to swing round in a circular direction. It is held in position by strong guy-ropes. Attached to the pole is a lattice girder which enables the heap to be increased in area.

When an endless wire-rope railway is used, the bridge follows as closely as possible the angle of repose of the sand. After reaching the desired height, tracks can be carried out horizontally without the necessity of a boom, and a flat surface can be made instead of a cone, which can be planted with grass to prevent the nuisance of sand being blown about.

Ore and coal are only stacked temporarily and are generally dumped from trestles on which cars run direct from the mine. A gangway may be built in the centre of where the ore pile is to be made, and is so constructed that when cars run in to draw the ore they can be filled from chinaman chutes.

Coal should not be stacked when it can be avoided. Some coals are more liable to spontaneous combustion than others. It is mostly due to the absorption and condensation of oxygen from the air. The smaller the coal, the larger the surface exposed to the air. Coal piles on account of their bulk are seldom under cover and are exposed to the weather. Moisture promotes chemical action. Coal heaps should not exceed 12 ft in height. Almost all coal will heat to some extent. To determine the safe limit which must not be exceeded, drive ⅝ in. iron rods into the heap at convenient places, allow them to remain there for a suitable time, then remove, and test the temperature by hand. If any part of the pile is over 140° F active measures must be taken to prevent further heating. When the coal in the interior of a pile ignites, the heat cokes a layer of coal round the glowing nucleus which will shed water and make it difficult to quench by playing water on the surface of the pile. Drive a pointed 2 in. pipe, the lower 3 or 4 ft of which are perforated with ⅛ in. holes, to the depth of the fire. Connect the pipe with a hose and force water into the heart of the fire.

CHEMISTRY

TABLE OF SYMBOLS, ATOMIC WEIGHTS, AND VALENCES OF THE ELEMENTS

Element	Symbol	Atomic Weight	Valence No.
Aluminium . .	Al	26·97	3
Antimony (Stibnium) .	Sb	121·76	3, 5
Argon	A	39·944	0
Arsenic	As	74·93	3, 5
Barium . . .	Ba	137·36	2
Beryllium (Glucinium) .	Be	9·02	2
Bismuth	Bi	209·00	3, 5
Boron	B	10·82	3
Bromine . . .	Br	79·916	1
Cadmium . . .	Cd	112·41	2
Calcium . . .	Ca	40·08	2
Cæsium . . .	Cs	12·00	2, 4
Carbon	C	140·13	3, 4
Cerium	Ce	132·81	1
Chlorine	Cl	35·457	1
Chromium . . .	Cr	52·01	2, 3, 6
Cobalt	Co	58·94	2, 3
Colimbium (Niobium) .	Cb	93·3	3, 5
Copper (Cuprum) . .	Cu	63·57	1, 2
Dysprosium . . .	Dy	162·46	3
Erbium . . .	Er	167·64	3
Europium . . .	Eu	152·0	3
Fluorine . . .	F	19·00	1
Gadolinium . . .	Gd	157·3	3
Gallium . . .	Ga	69·72	3
Germanium . . .	Ge	72·60	4
Gold (Aurum) . .	Au	197·2	1, 3
Hafnium (Celtium) .	Hf	178·6	—
Helium	He	4·002	0
Holmium . . .	Ho	163·5	3
Hydrogen . . .	H	1·0078	1
Indium	In	114·8	3
Iodine	I	126·92	1
Iridium . . .	Ir	193·1	3, 4
Iron (Ferrum) . .	Fe	55·84	2, 3
Krypton . . .	Kr	83·7	0
Lanthanum . . .	La	138·92	3
Lead (Plumbum) . .	Pb	207·22	2, 4
Lithium . . .	Li	6·940	1
Lutecium . . .	Lu	175·0	3
Magnesium . . .	Mg	24·32	2

TABLE OF SYMBOLS, ATOMIC WEIGHTS, AND VALENCES OF THE ELEMENTS—(*contd.*)

Element	Symbol	Atomic Weight	Valence No.
Manganese	Mn	54·93	2, 4, 6, 7
Mercury (Hydrargyrum)	Hg	200·61	1, 2
Molybdenum	Mo	96·0	3, 4, 6
Neodymium	Nd	144·27	3
Neon	Ne	20·183	0
Nickel	Ni	58·69	2, 3
Nitrogen	N	14·008	3, 5
Osmium	Os	190·8	2, 3, 4, 8
Oxygen	O	16·0000	2
Palladium	Pd	106·7	2, 4
Phosphorous	P	31·02	3, 5
Platinum	Pt	195·23	2, 4
Potassium (Kalium)	K	39·096	1
Praseodymium	Pr	140·92	3
Radium	Ra	225·97	2
Radon (Niton)	Rn	222·0	
Rhenium	Re	186·31	0
Rhoduim	Rh	102·91	3
Rubidium	Rb	85·44	1
Ruthenium	Ru	101·7	3, 4, 6, 8
Samarium	Sa	150·43	3
Scandium	Sc	45·10	3
Selenium	Se	79·2	2, 4, 6
Silicon	Si	28·06	4
Silver (Argentum)	Ag	107·88	1
Sodium (Natrium)	Na	22·997	1
Strontium	Sr	87·63	2
Sulphur	S	32·06	2, 4, 6
Tantalum	Ta	181·4	5
Tellurium	Te	127·61	2, 4, 6
Terbium	Tb	159·2	3
Thallium	Tl	204·39	1, 3
Thorium	Th	232·12	4
Thulium	Tm	169·4	3
Tin (Stannum)	Sn	118·70	2, 4
Titanium	Ti	47·90	3, 4
Tungsten (Wolfram)	W	184·0	6
Uranium	U	238·14	4, 6
Vanadium	V	50·95	3, 5
Xenon	Xe	131·3	0
Ytterbium	Yb	173·04	3
Yttrium	Yt	88·92	3
Zinc	Zn	65·38	
Zirconium	Zr	91·22	

Abbreviations

a.c., alternating current
amp., ampere(s)
amt., amount
app., apparatus
approx., approximate
aq., aqueous
at. wt., atomic weight
b.p., boiling point
B.Th.U., British thermal unit
C., centigrade
cal., calorie(s)
c.c., cubic centimetre
c.d., current density
chem., chemical (not chemistry)
cm, centimetre
coeff., coefficient
com., commercial
conc., concentrate
const., constant
c.p., candle power
crit., critical
cryst., crystalline (not crystallized)
cu m., cubic metre(s)
d., density
d.c., direct current
diam., diameter
e.m.f., electromotive force
est., estimate
evp., evaporate
F., Fahrenheit
ft, foot
g, gramme
gr. grain
h.p., horse-power
hr, hour
in., inch
insol., insoluble
kg, kilogramme
kw, kilowatt
l, litre(s)
lab., laboratory
lb, pound weight
m, metre
max., maximum
mech., mechanical
mg, milligramme
min., minimum
ml, millilitre
mol., molecule
m.p., melting point
N., normal
No., number
ppt., precipitate
qual., qualitative
quant., quantitative
r.p.m., revolutions per minute
satd., saturated
sec, second(s)
sep., separate
sol., soluble
soln., solution
sp. gr., specific gravity
temp., temperature
v, volt(s)
vol., volume
w, watt(s)
wt., weight

The common logarithm of the reciprocal of the hydrogen ion concentration, C_H or $[H]$, of the solution expressed in gramme-ions per litre. Thus—

$$P_H = \log \frac{1}{[H]}$$

If $[H] = 10^{-4}$ there will be 0·0004 grammes of ionized hydrogen present in a litre of solution, and $P_H = 4$. P_H is therefore the index of the concentration with the sign reversed.

Blow-pipe Analysis*

PRELIMINARY TESTS

EXP. 1. *Heat in a piece of hard glass tubing closed at one end*

(*a*) Gas and vapour are given off which are—

Colourless and Odourless	Colourless and Smelling	Coloured and Smelling
Water (H_2O) of crystallization, of hydration, or moisture. *Oxygen* (O) peroxides, nitrates, chlorates, bromates, and iodates. Recognized by a glowing chip burning more brightly when inserted in the tube. *Carbonic anhydride* (CO_2) many carbonates and oxalates. *Carbon monoxide* (CO) oxalic and formates (the latter carbonize).	*Sulphurous anhydride* (SO_2). *Sulphuretted hydrogen* (H_2S) thiosulphate salts and hydrous sulphides. *Ammonia* (NH_3) some ammonia salts.	*Nitric peroxide* (N_2O_4) reddish-brown; most nitrates and nitrites. *Iodine* (I) violet; certain iodides and iodates. *Bromine* (Br) brown; some bromides. *Chlorine* (Cl) greenish-yellow; certain chlorides.

* Adapted from J. Landauer's *Systematischer Gang der Löthrohr-analyse*.

(*b*) Sublimates form

White Sublimates	Black or Grey Sublimates	Coloured Sublimates
Ammonia salts. *Mercurous chloride* (Hg_2Cl_2) sublimes without previous fusion. *Mercuric chloride* ($HgCl_2$) before subliming. *Antomony-trioxide* (Sb_2O_3) fuses and sublimes in shining needles. *Tellurium dioxide* (TeO_2) fuses and sublimes to an amorphous mass. *Arsenic-trioxide* (As_2O_3) sublimes without melting to octahedral crystals.	*Arsenic* (As) metallic arsenic and many arsenical combinations give a metallic reflection. *Amalgam* and many quicksilver combinations give metallic globules.	*Sulphur* (S) hot, yellowish-brown; cold, yellow. *Antimony-trisulphide* (Sb_2S_3) hot, black; cold, reddish-yellow. *Arsenic-trisulphide* (As_2S_3)hot, brownish-red: cold, reddish-yellow. *Mercuric-iodide* (HgI_2) yellow, on rubbing turns red. *Mercuric sulphide* (HgS) black, on rubbing becomes red. *Selenium* (Se) reddish to black; powder dark red.

(c) Change of colour

Oxide	Hot	Cold
Zinc oxide (Zn)O	Yellow	White
Stannic oxide (SnO_2)	Yellowish-brown	Light yellow to white
Lead oxide (PbO)	Brownish-red	Yellow
Bismuth oxide (Bi_2O_3)	Orange-yellow	Lemon yellow
Mercuric oxide (HgO)	Red to black	Red (volatile)
Ferric oxide (Fe_2O_3)	Red to black	Red (non-volatile)
Mercuric iodide (HgI_2)	Red to yellow	Red

(*d*) Fusible

Alkaline salts

(*e*) Carbonizes

Organic substances

(*f*) Phosphorescence

Alkaline earths, earths, zinc oxide, tin oxide.

(*g*) Decrepitates

Alkaline chlorides, and many minerals.

Exp. 2. *Heat in a piece of hard glass tubing open at both ends.*

(*a*) Gas and vapour given off are

Sulphurous anhydride (SO_2) known by its characteristic smell; sulphur and metallic sulphides.

Selenium dioxide (SeO_2) smells like rotten radishes; selenium and metallic selenides.

(*b*) Formation of sublimates

Arsenic trioxide (As_2O_3) very volatile, a white sublimate is deposited some distance from the assay; arsenic and metallic arsenides.

Antimony trioxide (Sb_2O_3) white fumes, some volatilize and some sublime; antimony and antimony compounds.

Tellurium dioxide (TeO_2) white fumes sublime to colourless drops; tellurium and compounds of tellurium with metals.

Lead sulphate ($PbSO_4$)
Bismuth sulphate ($Bi_2(SO_4)_3$) } white, generally found below the assay; compounds of sulphur with lead: resp. bismuth

Exp. 3. *Heat on charcoal.*

(*a*) Fusibility

Fusible	Infusible
Alkalis and some of the alkaline earthy salts. Antimony, lead, cadmium, tellurium, bismuth, zinc, tin (easily fusible). Copper, silver, gold (fusible with difficulty).	Salts of the alkaline earthy metals, silica. Iron, cobalt, nickel, manganese, molybdenum, wolfram, platinum, palladium, iridium, rhodium, and osmium.

(b) Deflagrates

Nitrates and halogen salts

(c) Swells up

Expulsion of water, borates and alum

(d) Colouring of the flame, reduction of metals, and formation of incrustations will be mentioned under the heading of characteristic examinations.

CHARACTERISTIC EXAMINATIONS

Examination for bases.

EXP. 4. *Heat the substance with bicarbonate of soda on charcoal in the reducing flame.*

If one of the following group reactions takes place by itself, then the examination can be shortened in the following manner—

(a) The substance gives an incrustation . . .	Begin with	Exp.	4,	No.	1
(b) The substance gives a metallic bead without an incrustation	,,	,,	4,	,,	10
(c) The substance gives a grey or black residue	,,	,,	5,	,,	13
(d) The substance colours the flame, especially when moistened with HCl	,,	,,	7,	,,	32
(e) The substance leaves a white luminous residue . . .	,,	,,	8,	,,	43
(f) The substance is completely volatile	,,	,,	9,	,,	52

The formation of hepar is an indication that sulphides or sulphates are present. When testing for S, should not use a gas flame.

1. *White incrustation.* Very volatile; disappears with a light blue halo and propagates a garlic odour. *Arsenic*

Special test. When heated with cyanide of potassium and bicarbonate of soda in a small glass tube closed at one end, a steel-grey mirror is formed, insoluble in HCl.

2. *Reddish-brown incrustation.* Variegated like the eye of a peacock's feather, driven off by O. and R. flame without a coloured halo. *Cadmium*

Special test. The scraped-off incrustation when heated with sodium thiosulphate ($Na_2S_2O_3$) in a glass tube closed at one end is coloured yellow.

3. *Incrustation yellow when hot, white when cold.* Is luminous and not volatilized. *Zinc*

Special test. When heated with nitrate of cobalt solution it is coloured green. If Cd and Zn are both present the Cd incrustation is first formed, and later on that of the zinc.

4. *Steel-grey incrustation.* Disappears in the R. flame with a blue colour, and gives off the smell of rotten radishes. *Selenium*

5. *White incrustation with a dark yellow to red border.* Disappears in the R. flame with a green colour. *Tellurium*

Special test. If Te and Se are both present, transfer the incrustation to a test tube, moisten with a few drops of conc. H_2SO_4 and heat slightly, Te is soluble at once to a carmine red colour, while the dirty green colour of the Se first appears on heating more strongly.

A delicate test for Te even in the presence of Fe or Cu is to fuse the powdered ore with about 5 times its weight of KCy in a closed tube; a purple ring will form above the fusion.

6. *Bluish-white incrustation.* Volatile, driven off with the O. flame, and it disappears with a green colour in the R. flame.

Bead, white, brittle, and oxidizable. *Antimony*

Special test. The detached incrustation is dissolved in dilute HCl on platinum foil; when Zn is added the Sb is thrown down on the platinum and stains it black.

7. *Incrustation orange when hot, lemon-yellow when cold.* Driven off by O. and R. flame without colouring them.

Bead, reddish-white, brittle, and oxidizable. *Bismuth*

Special test. When treated on charcoal with a mixture of iodide of potassium and sulphur in the O. flame a fine red incrustation of iodide of bismuth is formed.

8. *Incrustation lemon-yellow when hot, sulphur-yellow when cold.* Driven off by O. and R. flame, colours the R. flame a fine blue.

Bead, white, malleable and oxidizable. *Lead*

Special test. Treat the assay with HNO_3, evaporate the excess of acid, add a drop or two H_2SO_4, and heat till white fumes are evolved; a white powder remains that is insoluble in H_2SO_4 acidulated water.

9. *Incrustation yellow when hot, white when cold.* Very small in quantity and found close to the assay, not volatile.

Bead, white, malleable and very oxidizable. *Tin*

Special test. Dissolve in HCl and precipitate the metallic tin as a grey spongy mass out of the acid solution by means of metallic zinc; this will not cling to platinum as Sb does.

10. *Bead white, malleable, and very bright.* In the strong O. flame, a reddish-brown incrustation is formed; in the presence of Pb or Sb it becomes carmine-red. *Silver*

Special test. Dissolve in HNO_3; on the addition of HCl a white curdy precipitate of AgCl is thrown down.

11. *Bead yellow.* Very bright, malleable, not oxidizable. *Gold*

Special test. When dissolved in aq. regia, and $SnCl_2$is added, purple of Cassius is formed.

12. *Bead red.* Malleable and oxidizable. *Copper*

Special test. See Nos. 13 and 39.

Exp. 5. *The assay is dissolved in a borax bead on platinum wire.*

(*a*) The bead is coloured by the O. or R. flame. No. 13.

(*b*) The bead is not coloured by either flame. Ex. 7, No. 32.

The colour of the bead is

In the Oxidizing Flame		In the Reducing Flame		
Hot	Cold	Hot	Cold	
(13) Green	Bluish-green	Colourless	Brown	*Copper*
(14) Blue	Blue	Blue	Blue	*Cobalt*
(15 Violet to black	Reddish-violet	Colourless	Rose-red	*Manganese*
(16) Violet	Reddish-brown	Yellowish-grey	Yellowish grey	*Nickel*
(17) Red to yellow	Colourless	Green	Bottle green	*Iron*
(18) Brownish-red to yellow	Colourless	Reddish-yellow	Green	*Uranium*
(19) Yellow	Colourless l.q. opaque	Brown	Brown and opaque	*Molybdenum*
(20) Brownish red to reddish yellow	Grass-green	Green	Emerald-green	*Chromium*
(21) Red	Colourless to yellow	Colourless	Colourless	*Cerium*
(22) Yellow	Greenish-yellow	Brownish	Emerald-green	*Vanadium*
(23) Yellow	Colourless to yellow	Yellow	Yellowish-brown	*Wolfram*
(24) Yellow	Colourless	Yellow to brown	Violet	*Titanium*

Remarks

As a grey, infusible powder remaining behind, we have iron, nickel, cobalt (magnetic), molybdenum, wolfram, and the metals of the platinum group. The first-named bodies can be tested by borax in Exp. 5, but the platinum metals cannot be well tested by blow-pipe reactions. Some combinations of the metals with Cl, I, Br and S, form white, not very characteristic incrustations without reduction of the metal. These incrustations must not be mistaken for the above reactions. The substances which give these incrustations will be otherwise determined in the course of these tests.

13. Special test. When heated with microcosmic salt and tin in the R. flame, the bead becomes red; if it is black, roast it on charcoal and get rid of Sb and Bi with boric acid in the O. flame.

14. Special test. The metal reduced on charcoal if rubbed on paper and treated with HNO_3 forms a red solution, on adding HCl and drying a green spot is developed, which on moistening with H_2O disappears.

15. Special test. Fusing with bicarbonate of soda and nitre on platinum foil gives a green mass.

16. Special test. If the metal reduced on charcoal is rubbed on paper and treated with HNO_3, a green solution is formed, which on the addition of Na_2Co_3 gives an apple-green spot.

17. Special test. Rub the metal reduced on charcoal on a piece of paper, treat with HNO_3 and a drop of HCl; when warmed over a flame a yellow spot is left; if moistened with potassium ferrocyanide (K_4FeCy_6) it turns a blue colour.

18. Special test. The microcosmic salt bead is in the O. flame, yellow when hot, yellowish-green when cold; in the R. flame a dirty green when hot, and a fine green when cold (distinction from Fe).

Insoluble combinations of uranium are smelted on platinum with $HKSO_4$, and the fused mass rubbed up with Na_2CO_3 moistened and the liquid absorbed by paper, which, when moistened with acetic acid, gives a reddish-brown spot with K_4FeCy_6.

19. Special test. By digesting with H_2SO_4 on platinum foil, the MoO_2 colours the acid a deep blue on the addition of alcohol or by breathing on it.

20. Special test. By smelting with bicarbonate of soda and nitre on platinum foil a yellow mass is obtained.

21. Special test. Cannot be well determined by the blowpipe.

22. Special test. Fuse with bicarbonate of soda and saltpetre, dissolve in water, acidulate with acetic acid, add $AgNO_3$, which gives a yellow precipitate.

23. Special test. The microcosmic salt bead is colourless in the O. flame both hot and cold; in the R. flame it is a dirty green when hot, and blue when cold; in the presence of iron it is a blood-red. (See No. 27.)

24. Special test. The microcosmic salt bead is colourless when both hot and cold in the O. flame; but in the R. flame it is yellow when hot, and violet when cold; in the presence of iron it is blood-red. (See No. 30.)

25. When more than one coloured oxide is present, double reactions take place, for instance—

In the Oxidizing Flame		In the Reducing Flame		
Hot	Cold	Hot	Cold	
Violet to blood-red	Brownish-violet	Yellow	Bottle-green	*Mn and Fe*
Plum-coloured	Plum-coloured	Bluish-green	Blue	*Mn, Fe, and Co*
Green	Greyish-blue	Bluish-green	Green	*Mn, Fe, Co, and Ni*
Yellowish-green	Green	Greenish-blue	Blue	*Fe, Co, and little Ni*
Violet-brown	Brown	Blue	Blue	*Co and much Ni*
Green	Light green, blue or yellow, according to quantity	. . .	. . .	*Fe and Co* *Fe and Cu* *Fe and Ni*

25A. Special test. Several borax beads, with some of the assay dissolved in them, are collected and reduced on charcoal, with the addition of lead. After a few blasts separate the borax (*a*) from the lead (*b*).

(*a*) Dissolve some of the old borax slag with fresh borax in a platinum wire loop.

(α) The bead is blue. *Cobalt*

(β) The bead in the O. flame is green when hot, blue when cold. *Iron and Cobalt*

(γ) In the O. flame violet to blood-red when hot, brownish-violet when cold; in the R. flame yellow when hot, bottle-green when cold; when reduced on charcoal with tin gives a vitriol-green colour. *Manganese and Iron*

(δ) In the O. flame the bead is plum-colour both hot and cold; in the R. flame is bluish-green when hot, and blue when cold. *Manganese, Iron and Cobalt*

(*b*) Drive off the lead from the lead button with boric acid on charcoal in the O. flame, and dissolve the residue in microcosmic salt.

(α) The bead in the O. flame is blue when cold; with tin on charcoal it is reduced to a red colour. *Copper*

(β) The bead in the O. flame is yellow when cold. *Nickel*

(γ) The bead in the O. flame is green when cold. *Copper and Nickel*

Exp. 6. *Decompose the substance with bi-sulphate of potash, acidify with HCl, and add a strip of zinc.**

The solution is coloured—

26. Blue, then green, and finally dark brown. (No. 19.) *Molybdic acid*
27. Blue, then copper-red. (No. 23.) *Tungstic acid*
28. Blue, then green, and finally violet. (No. 22.) *Vanadic acid*
29. Green. (No. 20.) *Chromic acid*
30. Violet. (No. 24.) *Titanic acid*
31. Blue, in strongly acid solutions brown. *Niobic acid*

Exp. 7. *The substance, held in clean platinum forceps or wire, is placed in the non-luminous flame.*

(*a*) The flame is coloured (especially after moistening with HCl or H_2SO_4). No. 32

(*b*) The flame is not coloured. Exp. 8, No. 43

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* Exp. 6 can be omitted when wolfram, vanadium, titanium, and niobium are not sought.

The colour of the flame is—

	By Itself	Through Blue Glass	Through Green Glass		
After moistening with H_2SO_4, bring substance into the flame for a short time only	32. Violet	Reddish-violet	Bluish-green	*Potassium*	
	33. Orange	Reddish-violet	Orange-yellow	*Potassium and sodium*	
	34. Orange	Invisible or faint blue	Orange-yellow	*Sodium*	
	35. Carmine-red	Violet-red	Invisible	*Lithium*	
Moisten again with H_2SO_4, dry, and then use the greatest heat	36. Yellowish-green	Bluish-green	Green	*Barium*	Ba, Ca, and Sr can be recognized when together by noting the different coloured flashes after the assay, moistened with HCl, has been brought into the flame.
	37. Yellowish-red	Greenish-grey	Siskin-green	*Calcium*	
	38. Carmine-red	Purple	Faint yellow	*Strontium*	

39. Green; after moistening with HCl, blue. *Copper*

Testing for acids

40. Yellowish-green, similar to the barium flame. *Molybdic acid*

Special test. Gives with borax the reaction of No. 19.

41. Yellowish-green (the salt is moistened with H_2SO_4). *Phosphoric acid*

Special test. When heated with Mg in a closed tube, and moistened with water, it smells of phosphoretted hydrogen.

42. Fine green (the salt is moistened with H_2SO_4). *Boric acid*

Special test. Heat on platinum with $CaFl_2$ and $HKSO_4$, when the intense green flame of boric fluoride is obtained.

Remarks

HCl and HNO_3 also produce green-coloured flames, but they are weak and rapidly disappear.

The flame colorations of the already recognized elements, As, Sb, Pb (blue), Zn (greenish-white), are removed by the employment of conc. H_2SO_4.

EXP. 8. *Moisten the substance with cobalt solution on charcoal, and heat strongly.*

43. Blue infusible mass. *Alumina*

Special test. Does not colour the flame as No. 41, and does not give a Si-skeleton with microcosmic salt.

44. Blue infusible mass. *Earthy phosphates*

Special test. Gives a yellowish-green coloured flame: see No. 41.

45. Blue infusible mass. *Earthy silicates*

Special test. Gives a Si-skeleton in the microcosmic salt bead.

46. Blue glass. *Alkaline borates*

Special test. Gives a fine green-coloured flame: see No. 42.

47. Blue glass. *Alkaline phosphates*

Special test. Gives a yellowish-green flame: see No. 41.

48. Blue glass. *Alkaline silicates*

Special test. In the microcosmic salt bead gives a Si-skeleton.

49. Flesh-coloured mass. *Magnesia*

50. Violet mass. *Zirconia*

51. Green mass. *Zinc oxide, tin oxide, antimony oxide, titanic acid* (already found)

EXP. 9. *Heat the substance with bicarbonate of soda in a glass tube closed at one end.*

52. Metallic sublimate, which, when rubbed, unites into globules. *Quicksilver*

Special test. Heat with $Na_2S_2O_3$ in a closed tube, black HgS is formed.

53. Odour of NH_3. *Ammonia*

Special test. With HCl fumes, forms a white cloud of AmCl.

Examination for acids

EXP. 10. *Heat the substance with bisulphate of potash in a ube closed at one end.*

(*a*)	It evolves a coloured gas.	No. 54
(*b*)	It evolves a colourless, smelling gas.	No. 60
(*c*)	It evolves a colourless and odourless gas.	No. 68
(*d*)	It gives no reaction.	Exp. 11, No. 71

54. Red fumes, with a smell of nitrous acid. *Nitric or nitrous acid*

Special test. If a strip of paper, saturated with a solution of $FeSO_4$, is pushed into the tube, it is coloured brown.

Nitrates deflagrate with an explosion, and flame when heated on platinum foil with powdered KCy.

55. Yellowish-green gas, which smells of chlorine. *Chloric acid*

Special test. The substance deflagrates on charcoal.

56. Violet fumes, turns starch-paste blue. *Iodine*

Special test. The substance, together with oxide of copper in a microcosmic salt bead, colours the flame a pure green.

57. The previous reaction takes place on the addition of $FeSO_4$. *Iodic acid*

Special test. The substance deflagrates on charcoal.

58. Reddish-brown fumes, colours damp starch paper yellowish-brown. *Bromine*

Special test. The substance, together with oxide of copper in a microcosmic salt bead, colours the flame a greenish-blue.

59. The same reaction. *Bromic acid*

Special test. The substance deflagrates on charcoal.

60. Fumes, which, with NH_3, form white clouds, and have the smell of *Hydrochloric acid*

To determine I, Br and Cl when associated together mix with MnO_2 and add acetic acid: heat until all the violet fumes of I are given off: then add HNO which liberates brown fumes of Br, also detected by the smell: finally add H_2SO_4 which drives off yellow fumes smelling of Cl.

Special test. The substance, together with oxide of copper in a microcosmic salt bead, colours the flame an intense blue.

61. Strong-smelling pungent gas, which etches glass. *Hydrofluoric acid*

Special test. Heat the salt in a borax bead with bisulphate of potash; the flame is coloured green.

62. Smell of sulphuretted hydrogen. *Sulphuretted hydrogen*

Special test. When heated in an inclined glass tube open at both ends, metallic sulphides give a smell of SO_2, and the fumes turn blue litmus paper red.

63. Smell of burning sulphur; no separation of sulphur. *Sulphurous acid*

64. The same reaction, but with the separation out of sulphur. *Thiosulphuric acid*

65. A pungent-smelling gas that brings tears to the eyes and makes lime-water cloudy. *Cyanic acid*

66. Smell of vinegar. *Acetic acid*

67. Smell of almonds. *Hydrocyanic acid*

68. A gas is driven off that makes lime-water turbid. *Carbon dioxide*

69. The gas burns with a blue flame. *Carbon monoxide*

70. The substance is carbonized. *Organic acids*

Exp. 11. *Heat the substance, that forms a hepar with bicarbonate of soda on charcoal, with caustic potash on platinum foil; place the whole in a vessel of water with a clean silver coin.*

71. The coin is not browned. *Sulphuric acid*

Special test. To recognize sulphuric acid in the presence of sulphides (No. 62), dissolve the substance in water acidulated with HNO_3, and precipitate the H_2SO_4 with $BaCl_2$. Insoluble sulphates are first boiled in a solution of carbonate of soda, filtered and acidulated.

The following have already been found—

72. Phosphoric acid (No. 41), boric acid (No. 42), silicic acid (No. 45).

Chemical Arithmetic (Bayley)

To find the Percentage Composition having the Formula given. Multiply the atomic weight of the element by the number of the atoms of the element there are in the molecule; multiply the number thus obtained by 100 and divide by the molecular weight of the compound.

To find the Empirical Formula of a Body from its Percentage Composition. Divide the percentage of each element by the atomic weight of that element to three places of decimals and divide all the numbers thus obtained by the lowest; if the quotients are not whole numbers, reduce them to their simplest relation in whole numbers, and to these whole numbers prefix the symbol to which each refers.

To find the Weight of a Substance required to yield, liberate, or produce a given Weight of a Substance. Write the equation expressing the chemical change: then—

Molecular weight of resulting substance × number of molecules involved	:	Quantity of resulting substance given	: :	Molecular weight of original substance × number of molecules involved	:	Weight of original substance required

TABLE FOR THE CALCULATION OF ANALYSIS

Let w be the weight of the analysed substance and a the weight of the determined constituent of the same; then the percentage $P = \frac{100 \times a}{w}$ or $\log P = \log a + 2 - \log w$.

When the required constituent is weighed in a different combination than that in which it occurs in the analysed substance (e.g. S of FeS_2 as $BaSO_4$), let f be the factor by which the determined compound must be multiplied; then $P = \frac{a \times 100 \times f}{w}$ or $\log P = \log a + \log f + 2 - \log w$.

GRAVIMETRIC FACTORS AND THEIR LOGARITHMS

Weight	Sought	Factor	Logarithm—10
Ag = 107·880			
AgCl	Ag	0·75263	9·87658
	Cl	0·24737	9·39334
Al = 26·97			
Al_2O_3	Al	0·52914	9·72357
As = 74·96			
As_2O_3	As	0·75748	9·87937
Ba = 137·37			
$BaSO_4$	Ba	0·58848	9·76973
	BaO	0·65702	9·81758
	H_2SO_4	0·42016	9·62342
Bi = 209·0			
Bi_2O_3	Bi	0·89700	9·95279
Ca = 40·7			
CaO	Ca	0·71465	9·85409
$CaSO_4$	Ca	0·29434	9·46885
	CaO	0·41187	9·61476
Co = 58·94			
CoO	Co	0·78650	9·89570
Cr = 52·01			
Cr_2O_3	Cr	0·68426	9·83522
Fe = 55·84			
Fe_2O_3	Fe	0·69940	9·84473
	FeO	0·89980	9·95415
K = 39·096			
K_2PtCl_6	K	0·16084	9·20639
	K_2O	0·19375	9·28723
Mg = 24·32			
$Mg_2As_2O_7$	AsO	0·63730	9·80434
$Mg_2P_2O_7$	Mg	0·21842	9·33928
	MgO	0·36211	9·55884
	P	0·27865	9·44506
Mn = 54·93			
Mn_2O_4	Mn	0·72026	9·85749
Mo = 96			
$(NH_4)_2MoO_4$	$PbMo_4$	1·8727	10·27247

Gravimetric Factors and their Logarithms—(*contd.*)

Weight	Sought	Factor	Logarithm—10
Ni = 58·69			
NiO	Ni	0·78578	9·89530
Pb = 207·20			
$PbCrO_4$	Cr	0·16092	9·20661
$PbSO_4$	Pb	0·68323	9·83457
Sb = 131·77			
Sb_2O_4	Sb	0·79188	9·89866
Ti = 48·1			
TiO_2	Ti	0·60050	9·77852
U = 238·17			
U_3O_8	U	0·84806	9·92843
W = 184·0			
WO_3	W	0·79310	9·89933
Zn = 65·38			
ZnO	Zn	0·80338	9·90492

Assay Ton Weights

The assay ton weight is a system made up from a comparison of the avoirdupois, troy and gramme weights, and is found to be extremely simple and useful, saving a vast amount of calculation and labour.

The unit of the system is the Assay Ton (A.T.) which equals 29·166 g and is obtained as follows—

1 lb avoirdupois equals 7000 troy grains.

2000 lb equals 1 short ton.

2000 × 7000 equal 14,000,000 troy grains in 1 ton avoirdupois.

480 troy grains equal 1 oz troy.

14,000,000 ÷ 480 equal 29,166 troy ounces in 2000 lb avoirdupois.

There are 29·166 milligrammes in one assay ton.

2000 lb is to 1 A.T. as 1 oz troy is to 1 milligramme: therefore if 1 A.T. of ore assays 1 milligramme of gold or silver, the ton contains 1 oz troy.

Table for the Conversion of Percentage into Cwt and Lb per Ton and into Lb per Cwt

Per Cent	Per Ton		Per Cwt	Per Cent	Per Ton		Per Cwt
	Cwt	Lb	Lb		Cwt	Lb	Lb
1	—	22·4	1·12	51	10	22·4	57·12
2	—	44·8	2·24	52	10	44·8	58·24
3	—	67·2	3·36	53	10	67·2	59·36
4	—	89·6	4·48	54	10	89·6	60·48
5	1	0·0	5·60	55	11	0·0	61·60
6	1	22·4	6·72	56	11	22·4	62·72
7	1	44·8	7·84	57	11	44·8	63·84
8	1	67·2	8·96	58	11	67·2	64·96
9	1	89·6	10·08	59	11	89·6	66·08
10	2	0·0	11·20	60	12	0·0	67·20
11	2	22·4	12·32	61	12	22·4	68·32
12	2	44·8	13·44	62	12	44·8	69·44
13	2	67·2	14·56	63	12	67·2	70·56
14	2	89·6	15·68	64	12	89·6	71·68
15	3	0·0	16·80	65	13	0·0	72·80
16	3	22·4	17·92	66	13	22·4	73·92
17	3	44·8	19·04	67	13	44·8	75·04
18	3	67·2	20·16	68	13	67·2	76·16
19	3	89·6	21·28	69	13	89·6	77·28
20	4	0·0	22·40	70	14	0·0	78·40
21	4	22·4	23·52	71	14	22·4	79·52
22	4	44·8	24·64	72	14	44·8	80·64
23	4	67·2	25·76	73	14	67·2	81·76
24	4	89·6	26·88	74	14	89·6	82·88
25	5	0·0	28·00	75	15	0·0	84·00
26	5	22·4	29·12	76	15	22·4	85·12
27	5	44·8	30·24	77	15	44·8	86·24
28	5	67·2	31·36	78	15	67·2	87·36
29	5	89·6	32·48	79	15	89·6	88·48
30	6	0·0	33·60	80	16	0·0	89·60
31	6	22·4	34·72	81	16	22·4	90·72
32	6	44·8	35·84	82	16	44·8	91·84
33	6	67·2	36·96	83	16	67·2	92·96
34	6	89·6	38·08	84	16	89·6	94·08
35	7	0·0	39·20	85	17	0·0	95·20
36	7	22·4	40·32	86	17	22·4	96·32
37	7	44·8	41·44	87	17	44·8	97·44
38	7	67·2	42·56	88	17	67·2	98·56
39	7	89·6	43·68	89	17	89·6	99·68
40	8	0·0	44·80	90	18	0·0	100·80
41	8	22·4	45·92	91	18	22·4	101·92
42	8	44·8	47·04	92	18	44·8	103·04
43	8	67·2	48·16	93	18	67·2	104·16
44	8	89·6	49·28	94	18	89·6	105·28
45	9	0·0	50·40	95	19	0·0	106·40
46	9	22·4	51·25	96	19	22·4	107·52
47	9	44·8	52·64	97	19	44·8	108·64
48	9	67·2	53·76	98	19	67·2	109·76
49	9	89·6	54·88	99	19	89·6	110·88
50	10	0·0	56·00	100	20	0·0	112·00

Example. A lead ore contains 60 per cent of lead, how much is that per ton?

Opposite 60 per cent see 12 cwt/t.

Normal Solutions

Normal solutions as used in volumetric analysis are so prepared that one litre at 16° C. shall contain the hydrogen equivalent of the active reagent weighed in grammes (H = 1): a decinormal solution (N/10) is one-tenth of the normal strength. Thus in the case of univalent substances, e.g. silver, iodine, sodium, hydrochloric acid, etc., the equivalent of the atomic (or in the case of salts, molecular) weights is identical. A normal solution of HCl must contain 36·45 g of the acid in a litre of fluid, and NaHO 40 g. In the case of a bivalent substance, e.g. lead, calcium, sulphurous acid, oxalic acid, carbonates, etc., the equivalent is one-half the atomic (or in the case of salts molecular) weight. Thus a normal solution of oxalic acid would be made by dissolving 63 g of the crystallized acid in distilled water, and diluting the liquid to one litre. In the case of trivalent substances, e.g. phosphoric acid, a normal solution of sodium phosphate would be made by weighing 358 ÷ 3 = 119·4 g of the salt, dissolving in distilled water, and diluting to one litre.

Reagent	Normal Solution	Quantity to be weighed so that 1 ml N.sol. = 1 per cent of Substance	Normal Factor
Crystallized oxalic acid ($H_2C_2O_4$, $2H_2O$)	63·024	6·302	0·06302
Sulphuric acid H_2SO_4	49·038	4·9038	0·049038
Hydrochloric acid HCl	36·465	3·6465	0·036465
Nitric acid, HNO_3	63·016	6·302	0·06302
Anhydrous sodium carbonate, Na_2CO_3	52·997	5·30	0·053
Ammonia, NH_3	17·0314	1·703	0·01703
Sodium hydroxide, NaHO	40·01	4·001	0·04001

When preparing solutions for volumetric analysis, one must remember that it is not the value of a reagent as expressed by its equivalent hydrogen-weight that must always be considered, but rather its particular reaction in any given analysis. For instance: tin is a quadrivalent metal, but when using stannous chloride as a reducing agent in the analysis of iron, the half and not the fourth of its molecular weight is required, as shown by the equation $Fe_2Cl_6 + SnCl_2 = 2FeCl_2 + SnCl_4$. In like manner with a solution of KM_nO_4

when used as an oxidizing agent, it is the available oxygen that has to be taken into account, therefore in making a normal solution one-fifth of its molecular weight (158 ÷ 5) = 31·6 g must be contained in a litre. (F. Sutton.)

The coefficient by which the number of ml of N. sol. used in any analysis is to be multiplied in order to obtain the amount of pure substance in the material examined.

Potassium permanganate $2KMnO_4$ = 31·606 for use as an oxidizing agent in an acid solution gives up 5 atoms of available oxygen, equivalent to 10 atoms of available hydrogen. Have a normal solution of potassium permanganate contain $\frac{316{\cdot}06}{10} = \underline{\underline{31{\cdot}606}}$.

$$\begin{aligned} 1 \text{ ml N/10 } KMnO_4 &= 0{\cdot}005584 \text{ g Fe} \\ &= 0{\cdot}007184 \text{ g FeO} \\ &= 0{\cdot}007984 \text{ g } Fe_2O_3 \end{aligned}$$

Gold and Silver

Pot Assay. Grind the ore so as to pass through an 80 mesh sieve. If any metallics remain behind, dissolve them in aqua regia; dilute with water; sprinkle over the powdered sample; dry at 100° C.; mix, and weigh 50 g. If the ore has much sulphides, arsenides or antimonides, roast sweet in a muffle furnace, stirring constantly to prevent clinkering. Should the ore be very fusible mix a little clean sand with the assay. If As or Sb is present re-roast with the addition of a little carbon to decompose the arsenates and antimonates formed.

Mix the assay with about 40 g litharge or red lead, 2–3 g argol or powdered charcoal, 30 to 40 g of sodium carbonate, and 10 to 20 g borax. The amount of the fluxes will vary with the composition of the ore. The argol or charcoal is to reduce the lead oxide, the lead collecting the precious metals on its way to the bottom of the crucible. The sodium carbonate combines with the silica present, while the borax combines with the bases. The pot is placed in a wind furnace and heated till the mass becomes pasty. It is kept at this temperature for about 10 min after which the temperature is raised. When the mass is fluid and all action has ceased, pour into an iron mould that has been coated with chalk or blacklead to prevent the assay from sticking to the sides; allow to cool, detach the lead button from the slag by hammering, cupel, weight bead, and part. If the lead button is brittle, it is due to the presence of As, Sb, or Zn; if hard it is due to Cu or Sb. If the lead button is too large, scorify it to a reasonable size before cupelling. If the ore or tailings is poor, scorify two or more buttons to a suitable size.

Scorification Assay. This method is suitable for rich silver ores or those containing much copper which would be reduced in a pot assay. If granulated lead cannot be bought it can be prepared by taking silver free red lead or litharge in the proportion of 300 g to 15 g powdered charcoal and reducing to lead in a clay crucible. Pour it into a wooden trough, coated with black lead, keeping the molten lead just moving till solidification begins, then shake violently. The lead will break up into fine particles which can be separated from the larger pieces by sieving. Take 5 g of the ore, mix with 30 to 60 g granulated lead, cover with a little borax glass, and heat in a hot muffle furnace till the slag covers the lead completely; then add 0·5 g powdered charcoal to clean the slag.

When the lead thus formed disappears pour into a mould, allow to cool, break off slag, square button by hammering for easy handling with tongs, cupel, weigh, add silver if necessary, part with nitric acid, weigh again, and calculate. The silver may be added by recupelling or by fusion under a blow-pipe on charcoal. Parting is performed by brushing any particles of bone ash off the bead, flattening with a hammer, then treat with dilute nitric acid 1 : 2 in a porcelain crucible, then with stronger acid, 1 : 1, pour off and wash with water.

The parted gold should be heated to redness before weighing. If much silver is present weaker acid is used to prevent the gold from breaking up into very fine particles which are liable to be poured off with the acid or wash water. When making up parting acids, add a few drops of silver nitrate in order to precipitate any chlorides present. Silver chloride thus formed may be allowed to settle.

Bullion Assay. Smelt the bullion so as to make its composition as uniform as possible. If very impure take a dip sample in a charcoal or clay spoon, otherwise allow to cool and take a drill sample. Weigh 0·5 g of the sample accurately. If the amount being weighed is a little too heavy, rub a shaving down on a fine file. Place in a cupel with 8 g of lead foil. Clean the bead, weigh and part, wash, dry, ignite, and weigh again. This gives the approximate value of the bullion, the difference between the weight of the bullion taken, and the gold and silver bead, being considered as base metal. (The difference between the gold and silver bead, and the parted gold, being taken as silver.) Make a "check" by weighing pure gold, silver, and copper to form a 0·5 g alloy of the composition of the bullion found by trial assay, and

TABLE FOR CONVERTING PERCENTAGES INTO TROY WEIGHT PER STATUTE TON

Percentage	Per Ton			Percentage	Per Ton		
	Oz	dwt	gr		Oz	dwt	gr
0·0001	0	0	15·68	0·06	19	12	0
0·0002	0	1	7·36	0·07	22	17	8
0·0003	0	1	23·04	0·08	26	2	16
0·0004	0	2	14·72	0·09	29	8	0
0·0005	0	3	6·40	0·1	32	13	8
0·0006	0	3	22·08	0·2	65	6	16
0·0007	0	4	13·76	0·3	98	0	0
0·0008	0	5	5·44	0·4	130	13	8
0·0009	0	5	21·12	0·5	163	6	16
0·001	0	6	12·8	0·6	196	0	0
0·002	0	13	1·6	0·7	228	13	8
0·003	0	19	14·4	0·8	261	6	16
0·004	1	6	3·2	0·9	294	0	0
0·005	1	12	16·0	1·0	326	13	8
0·006	1	19	4·8	2·0	653	6	16
0·007	2	5	17·6	3·0	980	0	0
0·008	2	12	6·4	4·0	1306	13	8
0·009	2	18	19·2	5·0	1633	6	16
0·01	3	5	8·0	6·0	1960	0	0
0·02	6	10	16·0	7·0	2286	13	8
0·03	9	16	0·0	8·0	2613	6	16
0·04	13	1	8·0	9·0	2940	0	0
0·05	16	6	16·0	10·0	3266	13	8

EXAMPLE. 500 gr of ore gave 0·044 gr gold, what is the yield per ton?

One hundred gr of the ore will give 0·044 ÷ 5 = 0·0088 gr; and,

	Per cent		Oz	dwt	gr	
According to table	0·008	=	2	12	6·4	
,, ,,	0·0008	=	0	5	5·44	
So	0·0088	=	2	17	11·84	per ton

cupel in company with two lots of 0·5 g of the bullion to be determined, each with 8 g of lead foil.

The loss of gold and silver in the bullion due to cupellation may be ascertained by comparing the weights of the assays after cupelling with that of the check. After parting, the loss or apparent gain of gold on the check known as "surcharge," and which should not exceed 0·0003 g, will show the amount to be added to or deducted from the gold weights to give the true amount present in the bullion. The true silver content is found by adding the cupellation loss to the ascertained weight of the bead and deducting the corrected fine gold. The difference between the combined fine gold and silver and the weight of bullion is taken as impurities. To part gold completely, the proportion of Ag to An should be 4 or 5 to 1. Of course, the weight of any silver added in order to form an alloy capable of being parted must be deducted from the total silver found. Before parting, the bead should be hammered flat, rolled into a fillet ⅔ in. long, annealed, then

TABLE SHOWING THE QUANTITY OF FINE GOLD IN ONE OUNCE OF ANY ALLOY TO ONE-EIGHTH OF A CARAT

Fine Gold			Carat Gold			Fine Gold			Carat Gold		
Oz	dwt	gr	Carats	gr	eighths	Oz	dwt	gr	Carats	gr	eighths
1	0	0	24	0	0	0	5	20	7	0	0
0	19	4	23	0	0	0	5	0	6	0	0
0	18	8	22	0	0	0	4	4	5	0	0
0	17	12	21	0	0	0	3	8	4	0	0
0	16	16	20	0	0	0	2	12	3	0	0
0	15	20	19	0	0	0	1	16	2	0	0
0	15	0	18	0	0	0	0	20	1	0	0
0	14	4	17	0	0	0	0	15	0	3	0
0	13	8	16	0	0	0	0	10	0	2	0
0	12	12	15	0	0	0	0	5	0	1	0
0	11	16	14	0	0	0	0	4·375	0	0	7
0	10	20	13	0	0	0	0	3·750	0	0	6
0	10	0	12	0	0	0	0	3·125	0	0	5
0	9	4	11	0	0	0	0	2·500	0	0	4
0	8	8	10	0	0	0	0	1·875	0	0	3
0	7	12	9	0	0	0	0	1·250	0	0	2
0	6	16	8	0	0	0	0	0·625	0	0	1

coiled into a "cornet" having the original bottom of the bead on the outside.

Gold in Blister Copper. Four lots of ¼ assay ton of the borings are weighed and placed in 3 in. scorifiers, mixed with 60 g granulated lead and covered with 30 g of the same: a little borax glass may also be added. Scorify twice, adding fresh lead the second time; then combine two buttons and continue scorifying with fresh addition of lead till all the copper is removed. The slags should be saved, and either scorified after mixing with powdered charcoal or argol, or run down in a crucible. The buttons from the slag and assay should be scorified together till of convenient size, then cupel, part, and determine the gold in the usual manner. As an extra precaution, the button from the blister copper may be cupelled first, the cupel then broken up and run down with the slags; the gold thus obtained being added to the first result.

Lead Assay

Standard solution: dissolve 10 g of finely divided molybdate of ammonia in a litre of water. The indicator solution should be freshly made by dissolving 0·1 g tannin in 30 c.c. water.

To determine the lead value of the ammonium molybdate, weigh out three lots of 0·3 g of pure lead and dissolve each in dilute HNO_3. When the solution is complete, add enough H_2SO_4 just to cover the bottom of the beaker, boil till heavy white fumes show that all the HNO_3 has been expelled. Cool, dilute, filter, leaving as much of the $PbSO_4$ as possible in the beaker. Put the filter paper back into the beaker with the precipitate and heat with a strong solution of ammonium acetate. When all the $PbSO_4$ is dissolved, dilute to about 150 c.c. and boil, then run in the ammonium molybdate solution from a burette until a drop of the mixture gives a bright yellow colour when placed on a spot of tannin solution on a white tile. The three titrations should agree closely, and their average will show how many c.c. of the molybdate solution are equal to 0·3 g of lead, and by calculation how much lead each c.c. of the solution equals.

Place 1 g of ore in a 300 c.c. Pyrex conical beaker. Add 10 c.c. HCl and heat gently, add 20 c.c. HNO_3 and heat until brown fumes are expelled and the ore is decomposed. Add 20 c.c. H_2SO_4 and take to strong fumes, cool and dilute with a little cold water, then add 150 c.c. of hot water, boil, and cool in water-trough. Filter through No. 5 Whatman filter paper, throw precipitate on to paper and wash four times with cold water. Place precipitate and paper in the original beaker with 20 to 30 c.c. of ammonium acetate and 50 c.c. of hot water. Boil for 15 min. Stir with a glass-rod to break up paper and residue, make bulk to 250 c.c. with hot water. Titrate with $(NH_4)_2MoO$ using tannic acid as outside indicator.

Zinc Assay

Solutions required—

Standard $K_4Fe(CN)_6$, 32·5 g/l. 1 c.c. = 0·01 g Zn.

$Fe_2 Cl_6$, 10 per cent solution.

Place 0·5 g in a 300 c.c. pyrex conical beaker. Add 10 c.c. HCl, cover with a watch glass and evaporate till nearly dry. Add 20 c.c. HNO_3. When brown fumes are expelled, allow to cool a little and add 1 g of powdered $KClO_3$. Evaporate to dryness carefully on a warm asbestos-covered plate, but avoid fusing. Allow to cool and add 20 c.c. hot water. Detach any adhering material from the beaker with a rubber tipped glass-rod. Add 5 g NH_4Cl and 20 c.c. NH_4OH, make bulk up to 100 c.c. and boil for 15 min. Cool in water bath, then add gradually 2 g sodium peroxide and boil for 10 min. Filter into a 600 c.c. conical beaker. Wash precipitate five times with a 10 per cent solution of NH_4Cl. Dissolve precipitate into the original beaker with as little HCl as possible, wash with hot water until free from Fe_2Cl_6. Precipitate with NH_4OH and boil for 10 min. Allow to cool, add 2 g sodium peroxide and boil for 10 min. Filter into original filtrate and wash five times with a 10 per cent solution of NH_4Cl. Bring filtrate to the boil and make acid with HCl, then add 1 c.c. in excess. Add 5 g granulated lead carefully and allow to boil until all copper is deposited on the lead. Remove from plate and wash copper and sides of beaker, and make bulk up to 400 c.c. with hot water. Add 3 g cream of tartar and 8 drops of Fe_2Cl_6, 10 per cent solution. Make solution alkaline with NH_4OH and add 5 c.c. in excess. Titrate with standard solution of $K_4Fe(CN)_6$. Glacial acetic acid is used as an outside indicator which gives a blue colour. A good plan is to divide the solution into three equal parts. One can be buretted roughly, the second more carefully, and the third exactly.

To standardize the $K_4Fe(CN)_6$ solution—

Place 0·25 g pure zinc filings in a 600 c.c. pyrex conical beaker, cover with a clock glass, add 20 c.c. cold water and just enough HNO_3 to dissolve the zinc. Place on asbestos-covered hot plate and evaporate carefully to near dryness.

Remove from hot plate, wash cover and sides of beaker, make bulk up to 100 c.c. Add 5 g NH_4Cl, make bulk up to 400 c.c. and heat to 70° C. Add 3 g cream of tartar and 5 drops of a 10 sol of Fe_2Cl_6. Make solution alkaline with NH_4OH, then add 5 c.c. in excess. Titrate with $K_4Fe(CN)_6$, using glacial acetic acid as outside indicator.

Copper Assay

Potassium Iodide Method. If the solution of the metal contains HNO_3 it is evaporated with H_2SO_4 till the former is expelled; or the HNO_3 is neutralized with Na_2CO_3 and acetic acid added. The sulphate solution must be neutral or only faintly acid. Excess of acetic acid does not matter, and therefore is always used, as all free mineral acids must be eliminated. Copper may be conveniently separated from a hot H_2SO_4 solution by sodium thiosulphate which gives a flocculent precipitate of sub-sulphide mixed with sulphur which filters readily and can be washed with hot water. As and Sb if present, are also precipitated, but Sn, Zn, Fe, Ni, Co, and Mn are not. On igniting the precipitate, most of the As and S are expelled, an impure oxide of copper being left.

For technical purposes a useful standard of sodium thiosulphate solution is one containing 40 g of the crystals in a litre of water; 1 c.c. equals approximately 0·01 g Cu. The thiosulphate solution should be checked occasionally, as its strength is apt to change on standing. Potassium iodide solution is made up of 75 g to 500 c.c. water. The starch solution should be freshly made.

To standardize the thiosulphate solution, take 1 g pure copper, dissolve in the least possible quantity of HNO_3 (sp. gr. 1·2), boil to expel nitrous fumes, dilute. Neutralize with Na_2CO_3 till a precipitate appears, and add acetic acid to the solution to clear it. The liquid is then made up to 1 litre, and a quantity equal to about 0·2 g placed in a beaker. Add 20 c.c. potassium iodide. The thiosulphate solution is now run in from a burette until the iodine is nearly removed. Starch is then added and the titration continued till the blue colour disappears. The addition of a drop of N/10 silver nitrate makes the end point clearer because the silver iodide formed counteracts the violet tinge of the precipitate.

Treat 0·5 g of powdered ore with 5 to 6 c.c. NHO_3 (sp. gr. 1·4). Evaporate nearly to dryness. Dissolve incrusted matter by heating with 5 c.c. strong HCl. Add 7 c.c. H_2SO_4. Heat till the H_2SO_4 fumes freely, cool, dilute with 25 c.c. H_2O to dissolve any anhydrous ferric sulphate, and filter. The copper is now precipitated on two stout pieces of sheet aluminium about 1½ in. square with alternate corners turned up and down for ¼ in. to prevent the pieces lying flat. When all the copper is deposited, decant the liquid, well wash the copper and foil with hot water, and when they are quite clean dissolve the copper in HNO_3. Boil to remove the lower oxides of nitrogen, evaporate to about 30 c.c., neutralize with a little Na_2CO_3 or ammonia, make slightly acid with acetic acid, cool to near room temperature. Add about 3 g potassium iodide or the equivalent of a saturated solution, and titrate as for standard.

The number of c.c. thiosulphate taken multiplied by the value of the reagent, gives the weight of the copper in the sample.

Electrolytic Method. If the sample is matte or blister copper weigh out 5 g of not larger than 20 mesh. Dissolve the copper in a little HNO_3 and H_2SO_4 in an evaporating dish. Cover with an inverted funnel and evaporate nearly to dryness on a sand bath. Dissolve residue in H_2O. Add a few drops of HNO_3 if necessary. Filter off residue, wash twice with water. Add 10 c.c. HNO_3 of 1·2 sp. gr. and dilute liquid till the volume is 150 c.c. There should be an amount of free HNO_3 present not exceeding 10 c.c. before electrolysis, since the strength is decreased by the formation of ammonia. If the solution becomes alkaline from this cause, the copper is deposited as a brown loosely adherent coating difficult to handle without loss. Sulphates may be present but not chlorides.

A current of 0·2 to 0·5 amp and an electromotive force of 2 to 3 volts are suitable. This may be obtained from 2 or 3 Daniell's cells; or the ordinary lighting circuit may be used if D.C.; but if A.C. a rectifier must be used. This can be made by placing a saturated solution of ammonium phosphate in four small beakers, and using rods of aluminium for anodes and strips of sheet lead for cathodes. The current is led from a small switchboard fitted with lamp connections to the rectifier as shown in the Figure, and from the rectifiers to the terminals to which the cones or cylinders on which the copper is deposited are attached. With 100 volt circuit, one lamp does for 2 g assays: two lamps in parallel for 4 assays, and so on. The cathode on which the copper is deposited is a platinum cylinder or cone. To determine if all the copper has been deposited, add a little water to the electrolyte and notice if the newly exposed surface of the cathode remains bright; or to a few drops of the electrolyte add some H_2S water; this will detect 0·000005 g or more of copper remaining in solution. The solution should not be too dilute, or the deposit will be spongy unless a very weak current is used. The more concentrated the solution, the stronger the current that can be used. The cathode is washed without breaking the circuit by leaving the bottom of it in the liquid. Finally, rinse with alcohol, dry high above a flame, cool in a desiccator, and weigh. The difference between the weight of the platinum cathode, and the same with the copper deposited on it, gives the weight of copper in the sample. The electrolysis requires at least eight hours at ordinary temperatures; but if the solution is kept at 70 to 80° C. 0·2 g is deposited in 60 to 80 min.

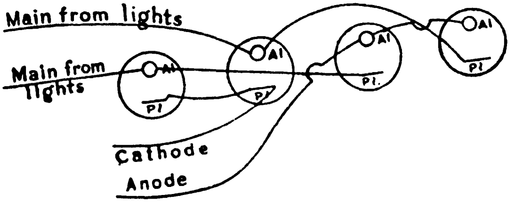


FIG. 75. CONNECTIONS

Tin Assay

The sample is generally that of concentrated or partly concentrated cassiterite. Crush fine and weigh out enough to leave about 10 g of tin oxide. If sulphides or arsenides are present roast, cool, and pan off carefully from one dish to another. If wolfram is present, treat with aqua regia for half an hour. Dilute, stir well, filter, wash several times with hot water, and then with ammonia to ensure the separation of tungstic acid. Dry, mix the residue with 40 g of KCy. Place 20 g KCy in the bottom of a clay crucible and add the above mixture to the top of it, then cover with another 20 g KCy. Place in a wind furnace, make it red hot for about half an hour. Tap the crucible to bring all the tin together into one button, pour into a conical iron mould, allow to cool, dissolve the slag in water and weigh the tin button. The tin button may be examined for purity by fracture or analysis.

Manganese Assay

Place 1 g powdered ore in a 300 c.c. beaker. Treat with 10 c.c. HCl. Heat gently. Add 10 c.c. HNO_3 to decompose sulphides and oxidize the iron present. Add 10 c.c. H_2SO_4 and heat to strong fumes of SO_3, cool, add 25 c.c. H_2O, and boil. Allow to stand hot until all anhydrous ferric sulphate has dissolved. Transfer the mixture to a 500 c.c. flask and add an emulsion of zinc oxide in excess to precipitate the iron. Dilute the contents of the flask to the mark with cold water, mix, and allow to settle. Draw off 10 c.c. of the clear liquid by means of a pipette and transfer to a 300 c.c. beaker. Heat the solution in the beaker to boiling. Add 2 or 3 drops of HNO_3 and titrate with a standard solution of potassium permanganate.

Slag Assays

Metallurgists generally require to know the composition of a slag within two or three hours in order to enable them to adjust the charge if necessary. Time is more important than chemical refinements. Volumetric methods are preferred to gravimetric as being quicker, by avoiding filtering, drying, incineration, and weighing. Standard solutions are made up so that 1 c.c. equals 1 per cent or ½ per cent of the substance assayed.

For iron blast furnace slags, it is often sufficient to determine the silica, alumina, and lime, which can be done in less than an hour and a half. Other constituents sometimes required are manganese, titanium, iron, magnesia, and sulphur.

For lead and copper blast furnaces, the most important determinations are for silica, iron, lime, alumina, and zinc; the less important being manganese, magnesia, baryta, and occasionally copper and lead.

The silica is returned as SiO_2, iron as FeO, lime as CaO, alumina as Al_2O_3, zinc as ZnO, manganese as MnO, magnesia as MgO, baryta as BaO.

If gas is not available for heating purposes, ignitions can be done in a muffle furnace. Evaporations can be carried out on an iron plate or sand-bath over an oil lamp. Glass-blowing, etc., can be done with the help of a blow-lamp.

A slag sample is taken by dipping an iron rod in molten slag and plunging it in cold water. The dry sample is ground to pass through an 80 mesh sieve. It can then be decomposed by aqua regia. If the slag to be assayed is already solidified, it will have to be fused with 5 or 6 g Na_2CO_3 in a platinum crucible in a muffle furnace for 10 min. The fusion is then dissolved in a casserole, after which it is treated as if it had been chilled.

A rapid, though rough method for determining the chief constituents of an iron blast furnace slag is as follows—

Weigh out 0·5 or 1 g of the finely-powdered chilled slag. Place in a small glass beaker. Add sufficient strong HCl to cover the bottom of the beaker and a few drops of HNO_3 to oxidize all the iron present. Evaporate to dryness. Take up with a little strong HCl. Heat gently, dilute with water, boil, filter off, wash, dry, ignite, and weigh as SiO_2 though when titaniferous iron is present, a small quantity of titanic acid will be in the residue, and if there is any baryta in the ore it will also be necessary to examine the silica for BaO.

To the filtrate from the above add ammonia till the solution is slightly alkaline, then add ammonium acetate. Boil at once and filter immediately. Wash, dry, ignite, and weigh as Al_2O_3, as the quantity of iron present is rarely sufficient to colour the alumina.

To the filtrate add HCl to make it distinctly acid. Oxidize with bromine and shake well for a few minutes. Add ammonia in excess. Boil briskly for several minutes. Filter, wash, dry, ignite, and weigh as Mn_3O_4 and calculate to MnO.

To the filtrate from the above, which is ammoniacal, add oxalate of ammonia. Boil well, filter, wash, dry, and ignite at a high temperature. Weigh as CaO.

The determination of magnesia can generally be left out, but should it be required, to the filtrate from the above add sodium phosphate. If the filtrate is too bulky, reduce it to a convenient bulk by evaporation. Allow to stand, if possible, over night in the cold. Filter, wash with cold water containing a little ammonia. Dry, ignite, weigh as $Mg_2P_2O_7$, and calculate to MgO.

To determine the sulphur present, weigh 1 g of the slag. Dissolve in aqua regia. Evaporate to dryness. Take up with a little HCl, add water and boil. Filter, wash. Add $BaCl_2$ to precipitate $BaSO_4$. Filter, wash, dry, ignite, weigh. Calculate the *sulphur* from the weight of $BaSO_4$ obtained.

For the determination of *iron*, weigh out 2 or 3 g of the slag. Add HCl and evaporate to dryness. Take up with a little HCl, dilute, boil, filter, wash well. To the filtrate add sufficient $SnCl_2$ to reduce the iron to the ferrous condition, and titrate with potassium bichromate.

To determine the titanic acid present in a slag, weigh 1 to 3 g of the slag. Fuse in a platinum crucible with eight times its weight of potassium bisulphate for 20 min. Allow to cool.

Detach from the crucible and grind to a paste with water in a small clean Wedgewood mortar. Transfer contents to a large beaker, dilute with water (all the liquids used must be cold), add 25 c.c. strong HCl. Allow to stand for three hours, stirring it from time to time. Filter, wash well. To the filtrate add ammonia till the solution is just alkaline and a slight precipitate is formed. Add 5 c.c. strong HCl, then add several crystals of sodium-thiosulphate. Boil for at least an hour, replacing the evaporated water from time to time if required. Filter, wash, dry, ignite, and weigh as TiO_2.

The following is the method employed by the Broken Hill Associated Smelters at Port Pirie, South Australia, for lead and copper blast furnace slags—

Solutions required. Standard $K_2Cr_2O_7$, 7 g l. 1 c.c. = 0·01 FeO.

Standard $KMnO_4$, 5·68 g l

1 c.c. = 0·0038 g MnO
1 c.c. = 0·005 g CaO.

Ammonium Phosphate 10 per cent, Sodium thiosulphate $Na_2S_2O_3$ 20 per cent.

Ammonium Acetate 1 pt ammonia, 2 pt acetic acid, 2 pt water.

Acid for dissolving slag kept in stock. One pt HNO_3, 3 pt HCl, 4 pt water.

Zinc Oxide which has been gently ignited in furnace and screened.

1. Place 1 g finely-powdered chilled slag in a 3½ in. casserole.

2. Add 25 c.c. of the aqua regia, evaporate gently to dryness, bake for a few minutes.

3. Take up with 15 c.c. HCl and 15 c.c. distilled water, boil for 10 min.

4. Filter through No. 597 filter paper into a 250 c.c. flask; wash with boiling water six times. Reserve filtrate for (6) and (II).

5. Residue is insoluble. Ignite in No. 1 porcelain crucible in hot muffle, cool, and weigh as *insoluble.*

6. Filtrate, cool in water-bath in reserve for (11)

7. Fuse insoluble in platinum crucible with 3 or 4 times its weight of anhydrous sodium carbonate for 10 min over blow-pipe.

8. Extract the melt with hot distilled water and HCl, in evaporating dish.

9. Take gently down to complete dryness on hot asbestos plate.

10. Powder finely with glass rod, add 20 c.c. HCl and 20 c.c. H_2O.

11. Warm till the salts are in solution. Filter into the flask containing the filtrate from (4) and wash six times with boiling water.

12. Place residue in No. 1 porcelain crucible. Ignite in muffle.

13. Cool, and weigh as *SiO_2*.

14. Cool the combined filtrates in flask and make up to the mark with distilled H_2O.

15. Transfer with pipette 100 c.c. into a 600 c.c. beaker.

16. Add 2 g ammonium chloride, 20 c.c. bromine water and boil for 10 min.

17. Add 5 c.c. bromine water and enough AmOH to ppt. the iron and manganese. Boil for 5 min.

18. Filter into a 600 c.c. beaker. Wash precipitate six times with boiling water, and reserve filtrate for (22).

19. Dissolve ppt. from filter paper into original beaker with HCl and H_2O, and add 10 c.c. HCl in excess.

20. Heat, add sufficient $SnCl_2$ drop by drop to reduce iron to ferrous condition, shown by the solution becoming colourless, then enough $HgCl_2$ to take up excess of $SnCl_2$. Make bulk up to 400 c.c. with cold water.

21. Titrate with $K_2Cr_2O_7$ for *iron*, using potassium ferricyanide as indicator on spot plate.

22. Filtrate from (18) boil and add 10 c.c. saturated solution of ammonium oxalate, boil 5 min, allow to settle 15 min.

23. Filter through No. 597 filter paper, wash twelve times with boiling distilled water.

24. Transfer filter paper and precipitate to beaker containing 400 c.c. of boiling distilled water and 7 c.c. of H_2SO_4.

25. Titrate with $KMnO_4$ to first pink tinge for *CaO*.

26. Transfer 50 c.c. from 250 c.c. flask into 600 c.c. beaker.

27. Add 5 c.c. HNO_3 and boil, then 10 g zinc oxide to form emulsion.

28. Make bulk to 400 c.c. with boiling distilled water.

29. Titrate with $KMnO_4$ to first pink tinge for *MnO*.

30. Transfer the remaining 100 c.c. in flask to 600 c.c. beaker. Add 30 c.c. ammonium phosphate 10 per cent solution.

31. Add ammonia cautiously until a slight ppt. forms.

32. Add just sufficient HCl to clear solution, then add 50 c.c. $Na_2S_2O_3$ solution.

33. Dilute to 300 c.c. with water, heat to boiling.

34. While boiling add 20 c.c. ammonium acetate solution. Boil 10 min.

35. Allow ppt. to settle, filter, wash six times with boiling water.

36. Open up filter, wash off ppt. with HCl and water into original beaker. Re-precipitate as described above.

37. Dry, ignite ppt., cool in desiccator and weigh as $AlPO_4$.

38. Factor for $Al_2O_3 = 0{\cdot}4185$.

39. The potassium dichromate is standardized with ferrous ammonium sulphate which contains one-seventh of its weight of Fe.

40. Duplicate assays are run on 1·4 g dissolved in cold distilled water.

41. The dichromate is carefully run in from burette till the spots of indicator (potassium ferricyanide) give no blue colour on a white plate.

42. The potassium ferricyanide 0·3 g to 300 c.c. is made up bi-weekly.

43. The potassium permanganate is standardized with ferrous ammonium sulphate as above.

44. In this case the end point is reached when the pink tinge no longer disappears.

Coal Analysis

All determinations, except that for moisture, should be made with dry coal.

Moisture. Weigh 1 g powdered coal in a watch-glass. Heat over a water-bath at 212° F. for about an hour. Allow to cool in a desiccator. Weigh. Repeat the heating till the weight is constant.

Ash. This is the mineral matter in coal after ignition. One gramme of dried coal is placed in a silica crucible and heated: first gently, and later strongly, stirring with a platinum wire till all the carbon has been oxidized.

Volatile Hydrocarbons. One gramme of the powdered coal is placed in a porcelain or platinum crucible which is covered, and then heated in a muffle furnace till the flame which shows on the top disappears: then raise the temperature to red heat for a minute or two. Allow to cool, then weigh. The loss equals the volatile hydrocarbon and harmful sulphur, and is returned in per cent.

Coke. This is the residue after driving off the volatile hydrocarbons.

Fixed Carbon. This is found by the difference between the weight of the coke and that of the ash.

Sulphur. The sulphur present may be harmful or harmless. It is the volatile sulphur which is important as indicating the presence of iron pyrites which causes corrosion of iron of boilers. To ascertain the total sulphur, mix 1 g finely-powdered coal with 7 to 8 g nitre, 4 g carbonate of potash or soda, and 16 g sodium chloride, all of which must be pure. Fuse gently in a platinum crucible placed in a muffle furnace till thoroughly white and fluid. Allow to cool by placing on a cold iron. Dissolve in hot water slightly acidulated with hydrochloric acid. Heat nearly to boiling, then add a solution of baric chloride in moderate excess. Allow to settle at a gentle heat. Decant through a filter paper. Wash the precipitate in the beaker till free from chlorine; finally transfer the precipitate to the filter paper. Dry, ignite the precipitate at a moderate red heat apart from the filter paper, and weigh.

$$\frac{\text{Wt. of } BaSO_4 \text{ ppt.} - \text{wt. filter ash} \times 13{\cdot}7}{100}$$

For the harmless sulphur boil 5 g finely-powdered coal in a solution containing 5 g pure sodium or potassium carbonate. Filter. Acidify with HCl; add $BaCl_2$. Allow to settle, dry, incinerate, weigh as $BaSO_4$ and calculate as above. The difference between this and the total sulphur gives the percentage of harmful sulphur.

Specific Gravity. Select a coarse homogeneous fragment of the coal. Place in a sling of silk thread. Clean off any dust. Suspend from one of the pans of a balance and weigh in the air. Thoroughly soak in distilled water for some time; brush off any adhering air bubbles and weigh in water.

$$\text{Sp. gr.} = \frac{\text{Weight of coal in air}}{\text{Diff. of wt. in air and water}}$$

Calorific Value. This is expressed in calories which is the amount of heat required to raise the temperature of 1 kilogramme of water 1° C. To convert 1 g coal into British thermal units (B.Th.U.) which is the amount of heat required to raise the temperature of 1 lb water 1° F. at or near 39·1° F., multiply by 1·8. The calorific value of coal is generally determined by a bomb calorimeter, but where the apparatus is not available, one may use Berthier's method. Weigh 1 g finely-powdered dry coal and mix it thoroughly with 40 to 50 g pure PbO. Place in a clay crucible, add a layer of 30 g more PbO, then a cover of borax glass. Lute on a lid. Heat gradually till fused, then raise to red heat for 10 minutes. Tap the crucible to collect all the lead at the bottom. Allow to cool. Then break out the reduced lead, and weigh it. To find the number of calories or heat units, multiply the weight of the lead reduced by 237·6.

The Specific Heating Power. Multiply the absolute heating power by the specific gravity.

The relative strength of a coke, which is important when required for use in a blast furnace, may be determined by mixing various quantities of sand with a given quantity of powdered coal, driving off the volatile hydrocarbons, and noting when it is crushed by a given weight.

Other points to be noted are, whether the coal is easily inflammable. Is it good for coking? If it burns with a large or small flame, smoky or luminous. Does it burn quietly or decrepitate? Does the coke burn for a long time or does it easily go out? The colour of the ash, red or white. It is the composition of the ash rather than its quantity that determines its fusibility, which is an important matter, especially in relation to mechanical stokers. Alumina is the most refractory constituent, but its fusing point, 2000° C., is lowered according to the amount of silica, alkalis, and iron present: but with many coals these do not lower the fusing point sufficiently to cause trouble, i.e. below 1400° C., with the exception of iron.

Amalgamation

For economical crushing, ore should be reduced in size by degrees in machines designed for each grade. Stamps crush from coarse to fine in one operation and therefore are not efficient, for if a stamp comes down on a piece of 2½ in. it does not descend with the same force as if the piece was smaller; and smaller pieces alongside of it do not receive a full blow; so when using stamps, the feed should be broken beforehand to ¾ or ½ in. Gravity stamps are suitable for inaccessible places, in the hands of inexperienced people, as they are easily understood and repaired, and have the advantage of being foolproof. A stamp proper generally weighs from 600 to 1200 lb and consists of the stem, disc, head, and shoe, and there are five such stamps in a battery. The order in which the stamps fall is such as not to drive the ore to one end of the mortar box where it would bank up if they were dropped in the order 1, 2, 3, 4, 5. A suitable order of fall is 1, 5, 2, 4, 3.

Important points in connection with stamps are their weight, the height of the drop, the number of drops of each stamp per minute, and the height of discharge, according to the class of ore to be treated, also the number of holes to the square inch and their shape. Each stamp requires from 1 to 8 gal of water per minute depending on the class of ore; the larger quantity being required for clayey ore which thickens the pulp. For rapid crushing, heavy, fast dropping stamps are used in narrow boxes with a shallow discharge. The depth of discharge is measured from the bottom of the screen to the top of the die. The screen should have a slightly forward inclination: if vertical the lower portion tends to wear out before the upper, and the pulp does not remain so long on the screen. If the lower part of the screen becomes worn out before the upper, it can be turned upside down. Some people have specially constructed mortars in which amalgamated copper plates are fixed. Others add about a teaspoonful (4 oz) of quicksilver to each box every two hours, but if the gold is coarse it sinks to the bottom anyhow by gravity and the sp. gr. of quicksilver is less than that of gold. When quicksilver is used in the box, the mortar should be roomy so as to give the quicksilver space not to get floured. A roomy box crushes more finely but discharges pulp more slowly than a narrow box. If the stamps are used for pulverizing only and not for amalgamation, the pulp should be expelled from the box as soon as reduced small enough, and therefore the mortar box should be narrow.

Amalgam is an alloy of quicksilver with another metal and is used in the metallurgy of gold and silver. Quicksilver is sold in wrought-iron flasks, but should be handled in enamelled vessels, porcelain, or glass, never in metal vessels such as zinc or tin which would amalgamate with it. In order to amalgamate, the quicksilver must be clean. Impure quicksilver, if caused to roll down an inclined surface of glass or paper, will do so in pear-shaped globules with tails which leave a blackish film behind. Only pure quicksilver forms approximately spherical globules.

The chief impurities in quicksilver are zinc and lead. These may be largely reduced by redistilling slowly at a low temperature. One plan is to distil the bulk of the quicksilver into one vessel, and the last 10 per cent into another, this containing most of the impurities. Another way is to distil the quicksilver into a vessel containing weak sulphuric acid, this will dissolve the zinc and lead which is distilled over with the quicksilver. Quicksilver if coated with a film will not amalgamate with gold because the two never come in contact.

When quicksilver is broken up into globules, it is said to be floured; if the globules will not coalesce, the quicksilver is said to be sickened. There are many substances which cause quicksilver to sicken, such as arsenic, native sulphur, acid sulphate of iron from the decomposition of copper or iron, or arsenical pyrites, but this is often corrected by the addition of quicklime: the presence of clay, talcose material in the ore, also oxides of iron or manganese promote a sickening effect on quicksilver. Grease from bearings must be prevented from finding its way into battery boxes, also exhaust steam used direct to warm battery water should not be used on account of the cylinder oil present, as grease sickens quicksilver, but this may be got rid of by adding an alkali such as lime which forms a soluble soap.

Potassium cyanide and sodium amalgam tend to produce a reunion of particles of sickened quicksilver as they reduce to the metallic form the oxides, sulphates and chlorides of the base metals that coat the quicksilver; but this action causes the metals to pass into the quicksilver. Iron amalgam is only formed when the metal is in a very fine state, such as when new shoes and dies are fitted to grinding pans. Copper from detonators finds its way into battery boxes and becomes amalgamated.

Silver amalgam is sometimes used on the surface of copper plates to be amalgamated. It is made from three parts of fine silver to one part quicksilver. About ½ oz is required for each square foot of surface to be amalgamated. Sodium amalgam is also used when amalgamating copper plates: it is made by heating quicksilver in a porcelain dish to 300° F., pieces of sodium, not larger than a pea, are dropped into this at a time. When the sodium touches the quicksilver, there is a slight explosion and a bright flash until about 3 per cent of sodium has been added when action becomes less pronounced. Then the amalgam is poured on to a plate to cool and solidify. When hard it is broken up and kept in a bottle under naphtha.

When cleaning quicksilver by distillation, the retort should never be more than half full, and it may be retorted under a layer of lime or iron filings. The iron filings decompose the sulphides and all sulphur and arsenic liberated are absorbed.

For copper plates use No. 15 B.W.G. The copper must be of the best quality and not rolled to such an extent as to make the surface bright and hard, otherwise it will not be porous enough to absorb the quicksilver and will have to be annealed by heating the plate from below until wood will ignite on the upper surface, taking care that the heat is uniform. It is then flattened by beating with a wooden mallet on a flat slab. The plates are secured to a wooden table with copper tacks across, and wooden battens down the sides. The plates should overlap each other like the slates of a roof. To prepare a copper plate, scour it with sand and wood ashes: the former mechanically to remove oxide of copper and the latter to remove any grease.

Acid should never be used to clean copper plates, because it is almost impossible to get rid of its effect, and a green scum is constantly rising through the quicksilver. Electro-silvered plates with 1 to 3 oz silver to the square foot are more easily cleaned. The plates are then washed with clean water and finally with a solution of ½ oz cyanide of potassium to one pint of water to prevent the copper from oxidizing. To amalgamate the copper some people rub in sodium amalgam; others use quicksilver with sand and ammonia chloride. Quicksilver is sprinkled on as long as the plate will absorb it, then left for, say, half an hour when silver amalgam is rubbed in with a piece of rubber belting and chloride of ammonia. Two to five square feet of amalgamated plate is allowed for each stamp.

In a well-set plate one square foot will hold one ounce of gold. The water required is just sufficient to clear the table. Too much water carries away fine gold. The object is to spread the water in a thin sheet over the plates. When there is ample water, the plates are given an angle of ½ to 1½ in./ft. If water is scarce, by setting the plates at an angle of 18° pulp can be used with the minimum of water (55 per cent water to 45 per cent sand).

Amalgam on the plates should not be too hard and dry. When they show signs of becoming dry, a little quicksilver should be sprinkled through a piece of canvas tied over the neck of a bottle containing quicksilver. Amalgam is scraped off the plates with a sharp-edged piece of rubber belting, or if very hard with a cold chisel, after which the plate is dressed with cyanide and quicksilver. How often the amalgam is removed depends on the richness of the ore treated.

Where quicksilver wells are used, they are semicircular, about 3 in. in width, sloping slightly towards one end, where there is a drain hole. The wells help to turn the water over as well to trap any amalgam and coarse gold, but they are not much good if there is a considerable amount of pyrites present as this collects on the surface, preventing contact with the gold and has to be skimmed off from time to time. The copper plates are not laid right up to the wells, for fear they should draw up quicksilver and make the plates too wet. Plates should have a locked cover, not only to prevent theft but also to prevent men from walking on them.

To steam plates, first clean off any sand with a jet of water, then fasten an air-tight cover over the plates, leaving a space between the cover and the plates of about 2 in. Steam is led into the top section under a pressure of 10 to 30 lb and spread out the full width of the table through a perforated pipe. The exhaust, which may contain quicksilver vapour, passes away through a pipe above the roof. The steam is allowed to act for about 20 minutes. When the cover is removed, men must take precautions against salivation. The softened amalgam is then scraped off with hard rubber, and the plates dressed in the usual manner.

If desired to scale a copper plate when it is no longer required, heat it over a charcoal fire till red hot, then plunge into water. This will scale off most of the gold, what is left can be chiselled off. The scales are treated with nitric acid to dissolve the copper, filtered, washed, dried, mixed with flux, and smelted.

Amalgam scraped off the tables is thinned with quicksilver in an enamelled dish of water and worked up to bring the impurities to the surface which are then sponged or washed off. The bright quicksilver is squeezed through wet chamois-leather, and the ball of amalgam left behind is retorted.

Where there is much amalgam to be retorted, cylindrical retorts some 5 ft long and 3 ft diameter made of cast iron are set in firebrick. The neck, which passes outside the furnace, is 2½ in. diameter and has water circulating round it to condense the quicksilver fumes. The amalgam is introduced to the retort in cast-iron trays.

In smaller works the ordinary oval-shaped retort is employed. The inside is coated with a solution of clay, or whitewashed, to prevent the gold from sticking to the retort. The amalgam is made up into balls of a convenient size to fit into the crucible in which the gold will be smelted. Each ball is wrapped in paper, which becomes charred in the retort and so prevents the balls from adhering together. The flanges on the body and lid of the retort are moistened and luted with clay: rings are slipped over corresponding lugs and are tightened gradually and finally tightly. The condensing tube is placed in a trough of water or has cold water played on it. The end of the tube does not dip into water for fear the water may be sucked up into the retort, should it be allowed to cool; but a damp rag is wrapped round the end of the condensing tube and connects with the water. When no more drops of quicksilver are given off, heat the retort to dull-redness for five minutes, then allow to cool for an hour or two; the dome must not be taken off while the retort is hot for fear of mercurial vapour.

The retorted gold, which does not look much like gold unless scratched, is melted in plumbago or salamanda crucibles which must be carefully annealed beforehand; a little flux of borax and bicarbonate of soda are added. The mould into which the gold is poured is warmed and wiped inside with a little oil. By partly cooling the bar under water and then withdrawing it while still hot, most of the slag will crack off; the rest may be chipped off, or the gold may be remelted in a clean crucible without flux.

Blanket Tables. If the amount of pyrites in the ore does not exceed 1 per cent it may be saved on blankets; but if more, too much time is occupied in washing the blankets. The tables are given an inclination of ¾ to 1½ in./ft; if the pitch is greater the blankets are liable to get ruffled, causing irregular patches of concentrates, and pyrites may be lost in the channels formed. To clean up blankets, turn off the supply of pulp, fold the upper end over the lower, then fold sideways. Unfold in a tank of water.

TABLE OF SIZES, ETC., OF BATTERY SCREENS

No. of Needle	Corresponding Mesh	Width of Slot (In.)	Thickness of Iron (Russian Gauge)	Thickness of Iron (American Gauge)	Weight per Square Foot (Lb)
5	20	$\frac{29}{1000}$	No. 14	No. 23¼	1·15
6	25	$\frac{27}{1000}$	No. 13	No. 24	1·08
7	30	$\frac{24}{1000}$	No. 12	No. 24½	0·987
8	35	$\frac{22}{1000}$	No. 11	No. 25	0·918
9	40	$\frac{20}{1000}$	No. 10	No. 26	0·827
10	50	$\frac{18}{1000}$	No. 9	No. 27	0·735
11	55	$\frac{16}{1000}$	No. 8	No. 28	0·666
12	60	$\frac{15}{1000}$	No. 8	No. 28	0·666

The Cyanide Process

Preliminary tests should be made to ascertain whether an ore is suitable for cyaniding. Most silver and gold ores are suitable, but there may be certain cyanicides present which in some cases can be removed. The size of the metal particles must be found out, for if too large they may take too long to dissolve, so it should have a preliminary treatment by amalgamation. If the particles are small and locked up in larger grains, the ore must be pulverized sufficiently small to free them. Should there be much copper and iron pyrites present, it might be advisable to concentrate the sulphides and sell them to custom works, or treat them by roasting, perhaps leaching, and then cyaniding. Oxidized copper minerals are objectionable on account of first consuming cyanide, and then by coating the zinc in the precipitating boxes. If the shavings in the first box are coated with lead nearly all the copper can be precipitated on them. Unoxidized copper pyrites are little affected by cyanide. Mispickel does not affect cyanidation, but pyrrhotite and antimonite precipitate gold almost as soon as it is dissolved. Charcoal, droppings of goats and rabbits on tailing heaps, and graphite in shales, also precipitate gold. Tellurides are decomposed by roasting and can then be cyanided. Graphite requires a high temperature to roast it, but it may be floated off, or made innocuous by oiling well with petroleum just before cyaniding.

Among cyanicides may be mentioned K_2S, Na_2S, sulphates and arsenates of iron and copper, Mg_2SO_4 and Ca_2SO_4. Partly decomposed pyrite or pyrrhotite by reacting with lime used as a protective alkali, forms insoluble calcium sulphate which gives trouble in the precipitation boxes. In such cases reduce the alkalinity or use pre-aeration. The presence of mercury from previous amalgamation results in a high consumption of zinc. The quality of the water, especially if it is obtained from the mine also effects cyaniding. Lead acetate or litharge is used to remove soluble sulphides from cyanide solutions, so the amount of soluble sulphides present and the quantity of lead acetate or litharge necessary to be added must be ascertained. The acidity of the pulp and the amount of lime required to neutralize it must also be determined. The amount of cyanide necessary to treat the ore, and the time required to effect a satisfactory extraction must be discovered. It is no use roasting ore containing antimonite as the mineral melts at a low temperature and entangles any gold present. To give the percentage of gold saved by cyaniding or amalgamation is not a good way to state the recovery. Eighty-five per cent from a two-ounce ore is bad: but 85 per cent obtained from a five-pennyweight ore is good. It is the loss of gold in the tailings that should be considered.

To test for KCy use a standard silver nitrate solution: that commonly used contains 13·04 g $AgNO_3$ per litre, so 1 c.c. equals 0·01 g KCy. Titrate till a cloudiness or opalescence is obtained. Some use a few drops of strong potassium iodide as an indicator.

To test for alkalinity or acidity, titrate with a standard acid or alkali. A few drops of phenolphthalein are added as an indicator till a pink colour disappears or appears, depending on whether the test is for alkalinity or acidity. Alkalinity is generally calculated in terms of NaHO, KHO, or CaO, and the acidity as HCy.

Determination of gold in a cyanide solution. Take 292 c.c. (10 assay tons) of the solution; place in a large beaker, add 5 g zinc shavings and 40 c.c. of a 20 per cent solution of lead acetate. Bring to a boil, place in a fume cupboard and add 40 c.c. commercial HCl. When action has nearly ceased, boil again. Pour off waste solution. Squeeze the wad of spongy lead, then place between filter paper and squeeze again hard. Fold the spongy lead in a piece of lead foil to increase its weight to about 10 g cupel, and weigh bead.

A quicker method. Measure 200 c.c. of the solution, add a few drops of strong KCy so that the solution shall contain about 1 per cent KCy. Add about a gramme of zinc dust. Heat the solution to boiling point. Filter off the cyanide solution. To the remaining zinc dust add about 20 c.c. dilute (10 per cent) sulphuric acid and dissolve all the zinc by gently warming. Filter off the zinc sulphate formed through the same filter paper. Dissolve the residual metals in 10 c.c. aqua regia (6 pt strong HCl, 2 pt strong HNO_3, and 6 pt distilled water) pass while boiling repeatedly through the same filter and collect the gold solution in a test tube. Cool. To the gold solution add a few drops of stannous chloride. If the solution is rich in gold, the purple of Cassius will appear directly: if it is poor, the colour will require a few minutes to become evident. The result is then compared with those obtained from known amounts of gold in a standard gold chloride solution, treated with the same amount of stannous chloride.

Sand is treated in vats by percolation. Slime, that which passes through a 200 mesh, is treated by agitation, mechanically or by compressed air. The strength of cyanide solution

(KCy or NaCy) for treating gold ores is between 0·05 to 0·25 per cent, while that for silver is between 0·25 and 0·50 per cent. Slime is agitated for about 24 hours, allowed to settle and the liquor decanted off: or the pulp is passed through vacuum filters, which may be stationary as in the case of Moore's filter, or revolving as in the Oliver and Merrill types. The volume of wash water should be at least 1 to 1½ times the volume of moisture held in the cakes.

In cyanide vats 28 cubic feet of space is allowed per ton of ore plus 9 to 12 in. in vertical depth for filter space. Dry sand weighs 78 lb/cu ft; wet sand 98 lb.

Zinc, aluminium, and charcoal are the chief precipitants used. Metallic zinc may be in the form of shavings or dust. Dust presents a larger surface. A good zinc dust should be fine and of uniform light-grey colour. Sp. gr. 6·8 to 7·1. Weight per cubic foot 141 lb. Screen test, 0·5 to 1·0 per cent on 100 mesh, 1·5 to 7·5 per cent on 200 mesh. Zinc oxide not more than 3 to 10 per cent. Good zinc dust should have a minimum of oxide. Oxide of zinc is not only inert as a precipitant, but being very soluble in cyanide tends to foul the solution. The presence of lead is valuable up to 5 or 6 per cent, the zinc-lead couple hastening the precipitation of the gold and silver from low-grade solutions. For low-grade gold solutions of weak cyanide and excess of alkali, 1 lb zinc dust will precipitate the gold out of 6 tons solution. In strong solutions 1 lb zinc dust is required for 2½ tons solution. When precipitating gold on zinc, there should always be an excess of an alkali to prevent a white precipitate of $Zn(OH)_2 . ZnO$. Thirty-six cu ft or 224 gal of water weigh one ton. Allow 1 lb zinc for 1 oz gold. Pass the gold solution through the precipitation box at the rate of 2 to 3 t/h. Two hundred and thirty gallons of liquid are required to saturate one ton of dry sand. Provide 0·25 cu ft zinc shavings for each ton of ore treated per 24 hours. One cubic foot of teased out zinc shavings weighs 3·6 lb. Use 130 to 150 tons of solution to 100 tons of sand. A trough for zinc shavings is 12 to 20 ft. long, 12 to 30 in. high and wide divided into from 6 to 8 boxes by baffle-plates. Zinc boxes are calculated empirically on the basis of the number of tons of solution that can be satisfactorily precipitated per cubic foot of zinc shavings in position per 24 hours.

When zinc or aluminium dust is used, it is agitated with the gold solution, and the slime separated in a vacuum press. Aluminium dust does not foul mill solutions like zinc. Excess of zinc in the gold slime is dissolved out with H_2SO_4 or HCl; the residue dried, and in some cases roasted, mixed with 30 per cent bicarbonate of soda and 60 per cent borax. The weight of the flux should be 18 to 35 per cent that of the precipitate.

Charcoal when used as a precipitant is made from soft wood. One hundred-weight of charcoal will precipitate 10 oz gold. It is good to use as a save-all, but is liable to loss while burning off, the ash being light and bulky is easily blown away.

TABLE FOR STANDARDIZING VAT SOLUTIONS

Per Cent	Lb	Oz	Per Cent	Lb	Oz	Per Cent	Lb	Oz	Per Cent	Lb	Oz	Per Cent	Lb	Oz
0·01	0	3	0·11	2	3	0·21	4	3	0·31	6	3	0·41	8	3
0·02	0	6	0·12	2	6	0·22	4	6	0·32	6	6	0·42	8	6
0·03	0	8	0·13	2	8	0·23	4	8	0·33	6	8	0·43	8	8
0·04	0	13	0·14	2	13	0·24	4	13	0·34	6	13	0·44	8	13
0·05	1	0	0·15	3	0	0·25	5	0	0·35	7	0	0·45	9	0
0·06	1	3	0·16	3	3	0·26	5	3	0·36	7	3	0·46	9	3
0·07	1	6	0·17	3	6	0·27	5	6	0·37	7	6	0·47	9	6
0·08	1	8	0·18	3	8	0·28	5	8	0·38	7	8	0·48	9	8
0·09	1	13	0·19	3	13	0·29	5	13	0·39	7	13	0·49	9	13
0·10	2	0	0·20	4	0	0·30	6	0	0·40	8	0	0·50	10	0

This table shows the number of ounces and pounds of cyanide of potassium necessary to make up a ton of solution from 0·01 to 0·5 per cent. It is more convenient to have an indicator on the solution tank marked in tons rather than in feet. One ton of water measures 36 cu ft.

The amount of cyanide used per ton of ore is between 2 and 5 lb KCy, but for concentrates may go up to as much as 20 lb. When pulp is agitated, the solution is from 0·05 to 0·25 per cent of KCy for gold, and 0·25 to 0·50 per cent for silver. NaCy contains more cyanogen per unit of weight than KCy, but its strength is quoted commercially in terms of KCy.

Slags

R in the following tables represents a radical that contains an atom oxygen.

Formulae for silicates of bases having the composition R O; e.g. CaO, MgO, BaO, FeO, MnO, and ZnO.

		Oxygen Ratio	
Subsilicate .	$4RO + SiO_2 = R_4SiO_6$	2 in base	: 1 in acid
Monosilicate .	$2RO + SiO_2 = R_2SiO_4$	1 ,,	: 1 ,,
Bisilicate . .	$RO + SiO_2 = RSiO_3$	1 ,,	: 2 ,,
Trisilicate .	$2RO + 3SiO_2 = R_2Si_3O_8$	1 ,,	: 3 ,,
Sesquisilicate .	$4RO + 3SiO_2 = R_4Si_3O_{10}$	2 ,,	: 3 ,,

Formulae for silicates of bases having the composition R_2O_3; e.g. Al_2O_3, Fe_2O_3, Mn_2O_3.

		Oxygen Ratio		
Subsilicate	. $4R_2O_3 + 3SiO_2 = R_8Si_3O_{18}$	2 in base	:	1 in acid
Monosilicate	. $2R_2O_3 + 3SiO_2 = R_4Si_3O_{12}$	1 ,,	:	1 ,,
Bisilicate .	. $R_2O_3 + 3SiO_2 = R_2Si_3O_9$	1 ,,	:	2 ,,
Trisilicate	. $2R_2O_3 + 9SiO_2 = R_4Si_9O_{24}$	1 ,,	:	3 ,,
Sesquisilicate	. $4R_2O_3 + 9SiO_2 = R_8Si_9O_{30}$	2 ,,	:	3 ,,

Each sesquesilicate can be broken up into a monosilicate and a bisilicate; for instance—

$$R_4Si_3O_{10} = R_2SiO_4 + 2(RSiO_3)$$
$$R_8Si_9O_{30} = R_4Si_3O_{12} + 2(R_2Si_3O_9)$$

If a silicate contains but one base it is called monobasic, but if two or more a double silicate or multibasic silicate.

Subsilicates are easily fusible, flow thinly, but consolidate quickly and break up in doing so. Their colour is usually dark, and they have a high specific gravity: formed mostly during the refining of metals. Subsilicates are only used for special purposes, e.g. when it is cheaper to lose some of the metal in the slag than go to the expense of flux and the necessary fuel to melt a larger quantity of more siliceous slag.

Monosilicates are not so readily fusible as the former, neither are they so thinly fluid, but they cool quickly and form crumbly pieces that don't hold together. Specific gravity 4·2 to 3·6.

Monosilicates eat into the furnace lining when not kept sufficiently cool, and on account of solidifying easily they entangle globules of metal. They also form deposits in the furnace, and if heated too strongly reduce iron, causing sows, thus shortening the life of a campaign. Monosilicates dissolve simple metallic sulphides of Fe, Zn, Ca, etc.

A monosilicate is used when converting copper matte.

Slags used in smelting lead ores are usually monosilicates, the bisilicates not being suitable as they carry away too much lead as a silicate; they are also more viscous and cause irregularities in the working of the furnace. The monosilicate, however, requires more base for fluxing the silica, and is, therefore, not always economical, so other slags are aimed at having oxygen ratios of 2 to 3, 4 to 5, and so on.

Sesquisilicates have a low melting point and run smoothly without forming a thick crust over their surface. Their specific gravity allows a complete and rapid settlement of the matte.

Bisilicates flow thickly, and can be drawn out into threads. They cool slowly. Sp. gr. 3 to 3·5. Bisilicates are often made because they are not too difficult to fuse, and do not corrode the acid lining of furnaces so easily as more basic slags do, and owing to their cooling more slowly they do not deposit so readily in the furnace—besides giving time for the metal to settle through them.

In copper smelting one generally aims for a bisilicate slag from the ore, but it may be necessary to vary this in order to obtain the required grade of matte, since this may be raised or lowered by adding more or less silica to flux off the iron that would otherwise pass into the matte. If the ore reduces with difficulty, flux it in such a manner as to give the metals time to reduce before the slag separates out. When concentrating a matte, make a sesquilicate slag; it works more slowly, giving the matte time to roast. A lead smelter who has to slag off a high percentage of zinc aims to make a slag low in silica and high in iron. One who is treating zinc in a reverberatory furnace must plan just the opposite.

In blast furnace work, if too high a copper matte is not formed, the slag may be sufficiently poor to be discarded, and the gases, which escape at a temperature of about 300° C., are relatively free from metals. The slag from copper converters is so foul that it has to be re-treated, and converter gases escape at a temperature of about 1500° C., which carry away large quantities of metals. The volatilization losses of Au and Ag from a converter may be increased by the long period required to blow too low a grade of matte.

Trisilicates flow slowly, cool gradually, and can also be drawn out into threads; they are porcelain-like in appearance. They require a very great temperature to form them, and are seldom used. Sp. gr. 2·94 to 2·57.

The choice of a slag depends on the object of the operation, the composition of the materials at hand, taking their cost into consideration, and the cleanliness of the slag. One wants to use the least possible amount of fluxes and fuel. An unnecessary amount of flux increases the quantity of slag and the loss of metals in it. Besides the heat-producing substances in a fuel, one must consider the ash which has to be slagged off, likewise impurities in the fluxes used. Many of the difficulties met with in smelting are due to an excess of fuel, e.g. the reduction of iron which forms sows and shortens the life of a campaign.

The monobasic silicates generally require more heat to form than double or multibasic silicates, and of the compound silicates those composed of monoxide bases are more difficult to fuse than a mixture of monoxide and sesquioxide bases.

Of the monobasic silicates those of—

Iron and manganese require	1789° C. to 1832° C.
Lime	2100° C. to 2150° C.
Baryta	2100° C. to 2200° C.
Magnesia	2200° C. to 2250° C.
Alumina	2300° C. to 2400° C.

Of double silicates those of—

Lime and alumina form at from	1918° C. to 1950° C.
Lime and magnesia form at from	2000° C.
Baryta and alumina form at from	2050° C.
Baryta and lime form at from	2100° C.

Slags already formed fuse at a lower temperature than that required for their formation.

Slag calculations are based on the ratio of the O in the bases to that of the silica.

When calculating a slag to contain BaO or MgO, it simplifies matters to bring these substances to their equivalent of CaO. This is done by multiplying the percentage of BaO by 0·368, and the percentage of MgO by 1·4. These figures are obtained as follows—

$$\text{Molecular weight of CaO } \frac{56}{152} = \underline{\underline{0{\cdot}368}} \qquad \text{CaO } \frac{56}{40} = \underline{\underline{1{\cdot}4}}$$

In like manner MnO is calculated to its equivalent of FeO.

The volatilization of sulphur in ordinary copper smelting may be as much as 75 per cent, leaving 25 per cent active sulphur.

Lime is generally added in the form of limestone; it is used to remove silica; it cleans the slag and lessens its specific gravity, allowing the matte to settle more perfectly. About 15 per cent CaO gives the best results for copper ore smelting when plenty of iron is available. A high lime slag makes it low in lead, except when the presence of zinc interferes and requires more iron. When required to slag off a small amount of sulphur, as in iron smelting, it may be got rid of as sulphide of calcium or barium. The sulphur may be considered as equal to half its own amount of silica—really 28 to 60.

Magnesia is generally added in the form of dolomite. Magnesia has 1·4 times the fluxing power of lime, and gives little trouble when used alone under 8 per cent, but in the presence of Zn or Al_2O_3 is objectionable. It makes the slag less fluid, and when high magnesia slags are formed must have at least 20 per cent iron.

Baryta is used as heavy spar, it makes a slag fluid but heavy; some of it enters the matte as BaS. Matte does not separate well from slags carrying barium.

Zinc decreases the fluidity of the slag and requires more fuel; by entering the matte it decreases its specific gravity and makes it more difficult to settle. Being volatile zinc causes a loss of silver and forms accretions; 10 per cent Zn is the maximum limit that should be present in a charge.

Alumina tends to purify a slag, but if over 10 per cent is apt to make the slag sticky. Used in the form of clay-slate or alumina-bearing rocks; employed when smelting ores rich in lime. Aluminous slags are generally slow running, and require a high heat to make them flow properly.

Iron and Manganese, in the form of oxide or carbonate, are used to flux siliceous ores.

Silica as quartz and natural silicates is used for removing excess of bases.

A wide range in the composition of slags is allowable in copper smelting, depending on the class of smelting employed, yet the proportion of silica to iron and lime will be found to correspond approximately to definite silicates.

One must determine whether it is cheaper to use a lime or iron flux, and whether conditions are such that it is better to make a concentrating or converting matte. If the matte is lower than, say, 35 per cent copper, it is generally better to concentrate it by resmelting, which serves at the same time to reduce the amount of As, Sb, Pb and Zn present. Matte suitable for converting generally ranges between 35 and 60 per cent of copper; if too high it is apt to chill while being transported from the furnace to the converter, and the slag from the ore is liable to be too rich in copper.

Calculating the Charge

The size of a charge depends on a combination of circumstances, e.g. the size of the furnace, and the relative amounts of the various ores to be smelted. Draw up a table as shown, noting the assays of the various ingredients in their proper column. Take the greatest weight from the chief ore, and proportion the other constituents according to their production, or, if that is capable of regulation, according to the probable requirements for matte and slag formation. It does not matter whether the total weights exceed or are less than those required for the charge, because they can be adjusted later when the relative proportions of the ores and fluxes have been ascertained. If several ores are to be smelted, blend those of small bulk to make a mixture so that there are only two classes of ore to deal with. If desired to blend two ores, one containing 30 per cent and the other 45 per cent, in order to obtain a blend of 35 per cent, then—

$$\begin{array}{l} 45 - 30 = 15 \\ 35 - 30 = 5 \end{array} \quad \frac{5 \times 100}{15} = \underline{\underline{33\cdot33}} \text{ parts by weight of the 45 per cent ore.}$$

The proportion of the 30 per cent ore required is 100 − 33·33 or 66·67 parts.

From the quantities assumed in the trial calculations and the assays, calculate the number of pounds weight of each constituent, and add those of the same kind together.

The proportion of sulphur burnt off varies with the furnace, the ore, the length of time the charge is in the furnace, the flux, and the pressure of the blast. Of the sulphur that is left most passes into the matte, while about 1 per cent goes to the slag. The amount of sulphur retained for the matte will determine the matte fall. The usual matte fall is from 10 : 1 to 15 : 1. Let us assume in our particular case that there is a loss of 70 per cent of the sulphur in the ore and concentrates by volatilization and in the slag, then 70 per cent of 156 lb equals 109 lb, leaving 47 lb sulphur available for matte making plus 13·8 lb in the slag and coke ash, from which there was little or no loss, making 60·8 lb altogether.

$$\begin{array}{lrcl} Cu_2 & 126 & = & \left.\begin{array}{l} 51\cdot2 \text{ per cent} \\ 22\cdot8 \text{ per cent} \end{array}\right\} 74 \text{ per cent} \\ Fe & 56 & = & \\ S_2 & 64 & = & 26\cdot0 \text{ per cent} \quad 26 \text{ per cent} \\ \hline & \underline{\underline{246}} & & \underline{\underline{100\cdot0}} \end{array}$$

$$\frac{74}{26} \; \frac{Fe + Cu}{S} = \underline{3}$$

The matte formed may or may not have the above composition, and actually all the copper in the matte is not

TRIAL METHOD

Material	Total wt. lb dry	Moisture	Total wt. lb. wt.	Fe + Mn		Si O₂		S		CaO + MgO		Cu	
				%	Lb	%	Lb	%	Lb	%	Lb	%	Lb
Ore .	1000	3	Fill this in after correcting for moisture	27	270	31	310	10	100	—	—	4	40
Concentrates	200	4		30	60	13	26	28	56	—	—	12	24
									156				
Old Slag	250	—		33	82 5	30	75	5	12·5	—	—	7	17·5
Limestone .	100	1		—	—	7	7	—	—	47	47	—	—
Ironstone .	70	1		61	42·7	9	6·3	—	—	—	—	—	—
	1620												
Coke 8%	130			1·2	1·5	10	13	1	1·3	2	2·6	—	—
					456 7		437·3		13·8		49·6		81·5

Desired Composition of Slag

Si O_2 . . .	42%
Fe O . . .	45%
Ca O . . .	4%
Other components .	9%
	100%

Matte

S . .	60·80 units	
Fe . .	104·02	,,
Cu . .	75·38	,,
	243·20	,,

	lb
Total S in ore and conc. .	156
Less 70% vol. and in slag .	156
	47
S in old slag and coke ash	13·8
S available for matte . .	60·8
Ratio between S and Fe + Cu in matte	3
Fe + Cu in matte	182·4

Slag formed 42 : 100 :: 437·3 = 1041

If loss of Cu in slag is 0·3%, then

$$\frac{1041 \times 0·3}{100} = 3·123 \text{ lb Cu}$$

	lb
Total Cu. .	81·50
Lost in slag	3·12
Cu available for matte .	78·38
Fe + Cu in matte .	182·40
Fe in matte	104·02
Total Fe .	456·70
Required for matte .	104·02
Available for slag .	352·68

Slag

Assume this to be 95% of total matte, which would then be 256 lb of 30·6% Cu

Si O_2 437·3 = 233·2 acid units oxygen
Fe O 453·4 = 100·7 basic ,, ,,
Ca O 49·6 = 14·1 ,, ,, ,,
940·3 or ratio

352·68 × $\frac{9}{7}$ = 453·4 FeO

Matte-fall

$$\frac{256 \times 100}{1620} = 15·8\ \%$$

combined as Cu_2S, but for the present we will take it that the sulphur constitutes about one-quarter the weight of the matte, so multiplying the matte-making sulphur by three will give the amount of Fe + Cu in the matte, viz. 182·4 lb.

To determine the quantity of slag formed—

As SiO_2 in assumed
slag : 100 :: lb SiO_2 in charge : weight of slag
42 : 100 :: 437·3 : 1041

Assume the loss of copper in the slag to be 0·3 per cent, then—

$$\frac{1041 \times 0{\cdot}3}{100}\ 3{\cdot}123 \text{ lb cu}$$

Deduct this from the 81·5 lb in the charge, we then have 78·38 lb copper from the 182·4 lb combined Fe and Cu which leaves 104·021 lb in the matte. If we consider the Fe, Cu, and S to form 95 per cent of the total matte, we have 256 lb matte containing 30·6 per cent Cu.

Note down the composition of the slag it is desired to make. If the available sulphur makes too low a matte, we may have to abandon the slag originally desired and increase the SiO_2 of the slag in order to reduce the S in the matte by smelting more slowly. This increase of SiO_2 can only go on to a stage when the slag becomes uneconomical.

The balance of the Fe not required by the matte passes into the slag, and is calculated as its equivalent of FeO.

Fe : FeO :: 56 : 72, $\frac{72}{56} = \frac{9}{7}$; therefore to convert Fe to FeO multiply the Fe by $\frac{9}{7}$.

The CaO in limestone, allowing for reasonable impurities, is approximately 50 per cent, therefore the required amount of lime has to be multiplied by 2 to bring it to limestone.

The combined weights of SiO_2, FeO, and CaO in the slag amount to 940·3, but these constituents only form from 91 per cent of the desired slag, so the total weight of the slag will be 91 : 940·3 : : 100 : 1033.

Total wt. of slag	:	total wt. of constituent	::	100	:	theoretical percentage of constituent
1033	:	437·3	::	100	:	42·3 per cent SiO_2
1033	:	453·4	::	100	:	43·9 per cent FeO
1033	:	49·6	::	100	:	4·8 per cent CaO
		940·3				91·0

If the theoretical result comes within 2 per cent of the desired slag, that is considered near enough, owing to variations in the ores, imperfect weighing, and other irregularities.

If over 2 per cent another trial calculation must be made with other figures.

The matte fall is—

Weight of charge : weight of matte :: 100 : x
1620 : 256 :: 100 : 15·8

Supposing the ore, concentrates, and old slags available form the commercial basis of the charge, and it is desired to use only a limestone flux, there would be no occasion to make a trial calculation, but the required amount of limestone could be obtained by a straight out calculation. A bisilicate slag requires to have twice the percentage of O on the silica or acid side as on the base side.

$$\begin{array}{llll} \text{Si } 28 & \text{Ca } 40 & \text{Fe } 56; & 72 : 60 :: 1 : 0{\cdot}833 \text{ FeO factor} \\ \text{O}_2\ 32 & \text{O } 16 & \text{O } 16; & 56 : 60 :: 1 : 1{\cdot}07 \text{ CaO factor} \\ \overline{60} & \overline{56} & \overline{72} & \end{array}$$

The available Fe for slag is $414 - 104 = 310 \times \frac{9}{7} = 398{\cdot}6$ FeO; multiply this by the factor $0{\cdot}833 = 332{\cdot}03$ SiO_2 satisfied.

The SiO_2 less that in the limestone and ironstone as shown in the table is 424 lb, deduct 332·03 satisfied by the FeO, this leaves 91·97 unsatisfied. To find the amount of CaO necessary to form a bisilicate, divide the SiO_2 in the limestone by 1·07, in our case 7 per cent equals 6·5. Subtract this from the 47 per cent CaO in the limestone, which leaves 40·5 per cent CaO available. The 91·97 lb SiO_2 to be fluxed with CaO is divided by the factor 1·07, which gives 85. As the available CaO in the limestone is 40·5 per cent, then—

40·5 : 85 :: 100 : 209·9 lb limestone in the charge.

Balling's Factors. To shorten tedious calculations, Prof. Balling has drawn up tables giving factors for multiplying assay values of the base and silica according to the class of slag required.

If the slag forming bases are multiplied by their corresponding factors in Table B, and the amounts of SiO_2 so found added up, this can be deducted from the amount of SiO_2 in the ore should that be in excess, in which case the balance of SiO_2 must be multiplied by the factor of the base in Table A with which it is intended to flux it. If the flux is lime, then as the lime is generally added in the form of limestone, $\frac{CaCO_3 = 100}{CaO = 56} = 1{\cdot}785$, the amount of lime required must be multiplied by 1·785 in order to ascertain the necessary amount of limestone.

TABLE A

For ascertaining the necessary amounts of Bases to convert given amounts of of Si O_2 into slag

One part by weight of Si O_2 requires—	Parts by weight of bases
For Monosilicates:	
CaO	1·86
MgO. . . .	1·33
Al_2O_3 . . .	1·14
FeO	2·40
MnO. . . .	2·36
For Bislicates:	
CaO	0·93
MgO. . . .	0·66
Al_2O_3 . . .	0·57
FeO	1·20
MnO. . . .	1·18
For Sesquisilicates	
CaO	1·24
MgO . . .	0·88
Al_2O_3 . . .	0·76
FeO	1·60
MnO. . . .	1·57

Ratio of molecular weight of bases to that of SiO_2;

$$\frac{2CaO}{SiO_2}\ \frac{112}{60} = 1{\cdot}86$$

TABLE B

For ascertaining the necessary amounts of Si O_2 to convert given amounts of Bases into slag

One part by weight of base requires—	Parts by weight of silica
For Monosilicates:	
CaO	0·535
MgO	0·750
Al_2O_3 . . .	0·873
FeO	0·416
MnO	0·422
For Bisilicates:	
CaO	1·070
MgO	1·500
Al_2O_3 . . .	1·747
FeO	0·833
MnO	0·845
For Sesquisilicates	
CaO	0·803
MgO	1·25
Al_2O_3 . . .	1·310
FeO	0·625
MnO	0·633

Ratio of molecular weight of SiO_2 to that of the base;

$$\frac{SiO_2}{2CaO}\ \frac{60}{112} = 0{\cdot}5357$$

If a number of calculations are to be made, time can be saved by the use of a graph which should be of a convenient size. C. O. Bannister uses squared paper with 10 divisions to an inch. The graph is constructed as shown in the accompanying diagram, where the bases are represented by a horizontal line, and the silica for mono- sesqui- and bi-silicates are shown in vertical scales.

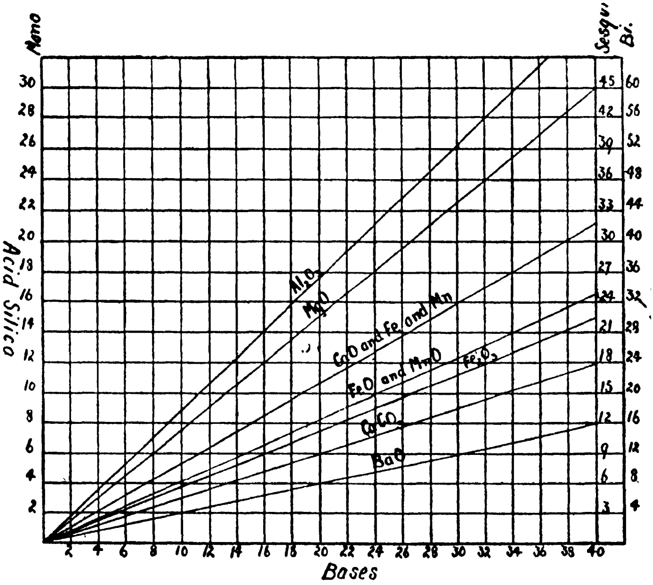


FIG. 76. THE BANNISTER GRAPH

From the point where the base and silica lines meet, draw lines to represent the different bases required in the formation of a slag at such an angle that the Al_2O_3 line cuts the horizontal line opposite 10 SiO_2 at 11·3, the MgO at 13·3, the CaO, Fe, and Mn at 18·6, FeO and MnO at 24, and BaO at 50·7. For convenience in calculating the fluxes required, lines for Fe_2O_3 cutting the 10 SiO_2 line at 26·6, and $CaCO_3$ at 33·3 are added. The atomic weights of Fe and Mn being so close to the molecular weights of CaO, the same line does for all of them.

The above distances are determined as follows—

$$\begin{array}{lll} \text{2CaOSiO} & Ca_2 & 80 \\ \text{Monosilicate of lime} & O_2 & 32 \end{array}\Big\} 112$$

$$60 : 112 :: 10 : 18{\cdot}6$$

$$\left.\begin{array}{ll} Si & 28 \\ O_2 & 32 \end{array}\right\} 60$$

Likewise $2MgO \,.\, SiO_2$ $\quad 60 : 80 :: 10 : 13{\cdot}2$
$2FeO \,.\, SiO_2$ $\quad 60 : 144 :: 10 : 24$
$2BaO \,.\, SiO_2$ $\quad 60 : 304 :: 10 : 50{\cdot}7$
$2Al_2O_3 \,.\, 3SiO_2$ $\quad 180 : 204 :: 10 : 11{\cdot}3$

Alumina having the composition of R_2O_3 requires $3SiO_2$ to form a monosilicate.

Fe_2O_3 is equal to 2FeO from a slag-making point of view; therefore, as the molecular weight of Fe_2O_3 is 160

$$60 : 160 :: 10 : 26{\cdot}6$$

and as $2CaCO_3$ is equal to 2CaO, the molecular weight of $2CaCO_3$ is 200, then—

$$60 : 200 :: 10 : 33{\cdot}3$$

Of the vertical silica lines, that of the bisilicate has double the number of spaces in the same distance as the monosilicate line; while the sesquisilicate has one and a half the number of spaces in the same distance as the monosilicate. All sesquisilicates may be resolved into a monosilicate and a bisilicate.

To use the diagram, it is advisable to employ set squares. By following the horizontal line cut by each base line above the percentage of any particular base given by analysis, one can ascertain the amount of SiO_2 required in order to form the particular class of slag desired according to the silica line read off. The excess of each base or SiO_2, as the case may be, over that necessary to form the slag considered advisable, can then be ascertained by reference to the analysis. By using the diagram, and noting the corresponding amount of SiO_2 or base necessary to satisfy the excess, the amount of flux required can be ascertained. One can also find out from the analysis of a slag whether it is of the type required, by comparing the amount of SiO_2 in the analysis with that calculated as necessary to combine with the bases present. In this way we can ascertain whether or not an excess of flux is being used.

Take an iron furnace charge supposed to be a mixture of iron ores which had to slag off—

Alumina	7·9
Lime	7·5
Magnesia	3·8
Silica	15·8

Referring to the graph we find—

7·9 pt alumina	require	6·95	pt silica
7·5 ,, lime	,,	4·00	,, ,,
3·8 ,, magnesia	,,	2·85	,, ,,
		13·80	

Therefore the excess of silica in the ore requiring to be fluxed is 15·8 − 13·8 = 2.

Comparing the graph method with the factors given by Balling we obtain practically the same results.

7·9 × 0·873 = 6·99 alumina
7·5 × 0·535 = 4·01 lime
3·8 × 0·750 = 2·55 magnesia

The nominal capacity of a copper blast furnace is 3¾ t/sq ft area at tuyère level, of ore and flux per 24 hours: but the capacity really depends on the nature of the ore, flux, and blast.

The water jackets require about 12 gal/min/sq ft of the jacket area. The temperature of the jacket should be about 85° C. A water jacket blast furnace for lead will smelt about half what a copper furnace will do, as the temperature has to be lower.

The quantity and pressure of air required for a blast furnace depends on the number and diameter of the tuyères, the amount of fines in the charge, and the height of the column. Formerly 12 oz per sq. in. pressure was allowed, but with the increase of the distance between the tuyères this pressure has been increased. As it is necessary to have an extra blower in case one in use has to be replaced, which would cause trouble if the proper supply was cut off, such extra blower, preferably of the piston type, could be used for any fluctuations in ordinary smelting in case of emergency. The cubic feet of air required per minute varies according to the size of the furnace from about 4000 to 90,000.

The amount of coke used is generally 14 per cent, but sometimes falls to 13 or rises to 17 per cent.

MISCELLANEOUS

Weights, Dimensions, and Properties of Various Articles

TABLE OF WEIGHT OF WATER (AT 62½ LB/CU FT) CONTAINED IN ONE FOOT LENGTH OF PIPES OF DIFFERENT BORES (*Trautwine*)

Bore In.	Water Lb	Bore In.	Water Lb	Bore In.	Water Lb	Bore In.	Water Lb
⅛	0·005320	6	12·25712	23	180·1116	62	1308·788
¼	0·021280	6¼	13·29983	24	196·1139	63	1351·347
⅜	0·047879	6½	14·38509	25	212·7972	64	1394·588
½	0·085119	6¾	15·51292	26	230·1615	65	1438·509
⅝	0·132998	7	16·68330	27	248·2067	66	1483·112
¾	0·191518	7¼	17·89625	28	266·9328	67	1528·395
⅞	0·260677	7½	19·15175	29	286·3399	68	1574·359
1	0·340476	7¾	20·44981	30	306·4280	69	1621·004
1⅛	0·430914	8	21·79044	31	327·1970	70	1668·330
1¼	0·531993	8¼	23·17362	32	348·6470	71	1716·337
1⅜	0·643712	8½	24·59936	33	370·7779	72	1765·025
1½	0·766070	8¾	26·06766	34	393·5897	73	1814·394
1⅝	0·899068	9	27·57852	35	417·0826	74	1864·444
1¾	1·042706	9¼	29·13194	36	441·2563	75	1915·175
1⅞	1·196984	9½	30·72792	37	466·1110	76	1966·587
2	1·361902	9¾	32·36646	38	491·6467	77	2018·680
2⅛	1·537460	10	34·04756	39	517·8633	78	2071·453
2¼	1·723658	10½	37·53743	40	544·7609	79	2124·908
2⅜	1·920495	11	41·19754	41	572·3394	80	2179·044
2½	2·127972	11½	45·02789	42	600·5989	81	2233·860
2⅝	2·346089	12	49·02848	43	629·5393	82	2289·358
2¾	2·574846	12½	53·19931	44	659·1607	83	2345·536
2⅞	2·814243	13	57·54037	45	689·4630	84	2402·396
3	3·064280	13½	62·05167	46	720·4463	85	2459·936
3⅛	3·324957	14	66·73321	47	752·1105	86	2518·157
3¼	3·596273	14½	71·58499	48	784·4557	87	2577·060
3⅜	3·878229	15	76·60700	49	817·4818	88	2636·643
3½	4·170826	15½	81·79925	50	851·1889	89	2696·907
3⅝	4·474062	16	87·16174	51	885·5769	90	2757·852
3¾	4·787938	16½	92·69447	52	920·6459	91	2819·478
3⅞	5·112453	17	98·39744	53	956·3958	92	2881·785
4	5·447609	17½	104·27064	54	992·8267	93	2944·773
4¼	6·149840	18	110·31408	55	1029·9386	94	3008·442
4½	6·894630	18½	116·52776	56	1067·7314	95	3072·792
4¾	7·681980	19	122·91168	57	1106·2051	96	3137·823
5	8·511889	19½	129·46583	58	1145·3598	97	3203·535
5¼	9·384358	20	136·19022	59	1185·1954	98	3269·927
5½	10·299386	21	150·14972	60	1225·7120	99	3337·001
5¾	11·256973	22	164·79017	61	1266·9096	100	3404·756

The weight of water in one foot length of any bore = inner diameter^2 × 0·339521.

TABLE OF APPROXIMATE PROPORTIONS AND WEIGHTS OF 9 FEET LENGTHS OF CAST-IRON PIPES OF VARIOUS SIZES

Diameter of Bore	Thickness of Metal	Diameter of Flange	Thickness of Flange	Diameter through Bolt-holes	Size of Holes	Number of Holes	Weight		
In.	In.	In.	In.	In.	In.	In.	Cwt	Qr	Lb
2	⅜	6½	9/16	4¾	⅝	4	0	3	0
3	⅜	7½	⅝	6	⅝	4	1	0	3
4	½	9½	¾	7¾	¾	4	1	3	5
5	½	10½	⅞	8¾	¾	4	2	1	12
6	⅝	12	⅞	10	⅞	4	3	2	1
7	⅝	14	1	11¾	⅞	6	4	3	17
8	¾	15	1	12¾	1	6	5	2	9
9	¾	16½	1 1/16	14¼	1	6	6	1	12
10	¾	17½	1⅛	15½	1	6	7	0	0
11	⅞	19	1 3/16	16¾	1	6	8	3	24
12	⅞	20	1¼	17¾	1⅛	6	9	3	5
13	⅞	21	1¼	18¾	1⅛	8	10	2	0
14	⅞	22	1¼	19¾	1⅛	8	11	0	26
15	⅞	23	1¼	20¾	1⅛	8	12	0	25
16	⅞	24½	1 5/16	22	1¼	8	12	3	8
17	⅞	25½	1 5/16	23	1¼	8	13	2	17
18	1	26½	1⅜	24	1¼	8	16	1	15
19	1	28	1⅜	25	1⅜	8	17	2	13
20	1	29	1⅜	26	1⅜	8	18	0	26

Table of the Weight of 100 Feet of Wire in Pounds, when the Size is measured by the New Standard Wire Gauge of 1884

Size by Gauge	Diameter in Inches	Copper Wire	Brass Wire	Steel Wire	Iron Wire
No.		Lb	Lb	Lb	Lb
1	0·300	26·8	25·7	24·0	23·4
2	0·276	22·5	21·6	20·1	19·6
3	0·252	18·8	18·1	16·8	16·4
4	0·232	15·8	15·2	14·3	13·8
5	0·212	13·4	12·7	11·9	11·6
6	0·192	10·9	10·5	9·7	9·5
7	0·176	9·2	8·8	8·2	8·0
8	0·160	7·6	7·3	6·8	6·6
9	0·144	6·1	5·8	5·4	5·3
10	0·128	4·99	4·65	4·34	4·23
11	0·116	4·00	3·80	3·55	3·46
12	0·104	3·22	3·08	2·87	2·80
13	0·092	2·48	2·37	2·22	2·16
14	0·080	1·91	1·83	1·70	1·66
15	0·072	1·53	1·46	1·36	1·33
16	0·064	1·22	1·16	1·08	1·06
17	0·056	0·92	0·88	0·82	0·80
18	0·048	0·69	0·66	0·62	0·60
19	0·040	0·46	0·44	0·41	0·40
20	0·036	0·38	0·36	0·34	0·33

If any less quantity than 100 feet is required, multiply the weight of 100 feet as found in the table by the number of feet wanted, and divide by 100 thus—

Required the weight of 50 feet of No. 16 copper wire.

100 feet of No. 16 gauge copper wire weighs 1·22 lb; therefore 100 : 1·22 :: 50 : required weight = 0·61 lb.

TABLE OF THE WEIGHT OF A SQUARE FOOT OF VARIOUS METALS

Thickness by Gauge	Thickness in Inches	Lead	Copper	Gun-metal	Brass	White Metal	Steel	Wrought Iron	Zinc
		Lb	Lb	Lb	Lb	Lb	Lb	Lb	Lb
7 — 0	0·500	29·59	22·87	22·68	21·86	20·88	20·40	19·87	18·72
6 — 0	0·464	27·66	21·18	21·04	20·28	19·36	18·93	18·44	17·37
5 — 0	0·432	25·74	19·75	19·58	18·87	18·02	17·62	17·16	16·19
4 — 0	0·400	23·67	18·26	18·14	17·45	16·68	16·32	15·89	14·97
3 — 0	0·372	22·17	17·01	16·86	16·28	15·52	15·18	14·78	13·94
2 — 0	0·348	20·74	15·92	15·78	15·21	14·52	14·20	13·83	13·05
1 — 0	0·324	19·31	14·81	14·68	14·16	13·51	13·22	12·87	12·14
1	0·300	17·75	13·70	13·60	13·08	12·51	12·24	11·92	11·23
2	0·276	16·45	12·63	12·52	12·07	11·51	11·26	10·97	10·35
3	0·252	15·03	11·53	11·43	11·02	10·52	10·29	10·02	9·45
4	0·232	13·83	10·61	10·57	10·14	9·68	9·47	9·22	8·70
5	0·212	12·64	9·70	9·67	9·27	8·85	8·66	8·43	7·95
6	0·192	11·44	8·78	8·75	8·40	8·01	7·84	7·63	7·20
7	0·176	10·29	7·90	7·87	7·55	7·20	7·04	6·86	6·48
8	0·160	9·54	7·32	7·29	7·00	6·68	6·53	6·36	6·00
9	0·144	8·58	6·58	6·56	6·30	6·00	6·13	5·72	5·40
10	0·128	7·62	5·85	5·83	5·58	5·33	5·22	5·08	4·80
11	0·116	6·91	5·31	5·29	5·07	4·84	4·73	4·61	4·35
12	0·104	6·20	4·75	4·73	4·54	4·34	4·24	4·13	3·89

TABLE OF THE WEIGHT OF A SQUARE FOOT OF VARIOUS METALS—(*contd.*)

Thickness by Gauge	Thickness in Inches	Lead	Copper	Gun-metal	Brass	White Metal	Steel	Wrought Iron	Zinc
		Lb	Lb	Lb	Lb	Lb	Lb	Lb	Lb
13	0·092	5·49	4·21	4·17	4·03	3·84	3·76	3·66	3·45
14	0·080	4·77	3·66	3·63	3·50	3·34	3·26	3·18	3·00
15	0·072	4·30	3·30	3·26	3·14	3·00	2·94	2·86	2·70
16	0·064	3·81	2·92	2·89	2·80	2·66	2·60	2·54	2·40
17	0·056	3·21	2·46	2·44	2·35	2·25	2·19	2·14	2·02
18	0·048	2·86	2·20	2·18	2·10	2·00	1·96	1·91	1·80
19	0·040	2·38	1·83	1·81	1·75	1·67	1·63	1·59	1·49
20	0·036	2·14	1·64	1·63	1·57	1·50	1·47	1·43	1·35
21	0·032	1·92	1·47	1·46	1·41	1·34	1·31	1·28	1·24
22	0·028	1·66	1·28	1·26	1·22	1·16	1·14	1·11	1·05
23	0·024	1·43	1·09	1·08	1·04	1·00	0·97	0·95	0·89
24	0·022	1·30	1·00	0·99	0·96	0·91	0·89	0·87	0·82
25	0·020	1·18	0·91	0·90	0·87	0·83	0·81	0·79	0·74
26	0·018	1·06	0·82	0·81	0·78	0·74	0·73	0·71	0·67
27	0·0164	0·97	0·75	0·74	0·71	0·68	0·67	0·65	0·62
28	0·0148	0·87	0·66	0·65	0·64	0·61	0·60	0·58	0·54
29	0·0136	0·81	0·62	0·61	0·60	0·57	0·55	0·54	0·50
30	0·0124	0·75	0·58	0·57	0·55	0·52	0·51	0·50	0·47

EXAMPLE. Find the weight of a sheet of copper 2½ ft × 4 ft × 0·128 in.

2½ ft × 4 ft = 10 sq ft; 0·128 in. = No. 10 by gauge; 1 sq ft of No. 10 copper sheet weighs 5·85 lb; therefore 5·85 lb × 10 sq ft = 58·5 lb. Ans.

TABLE GIVING EQUIVALENT VALUES PER LB, CWT, AND TON

Per Lb	Per Cwt		Per Ton			Per Lb	Per Cwt		Per Ton		
d.	*s.*	*d.*	£	*s.*	*d.*	*d.*	*s.*	*d.*	£	*s.*	*d.*
¼	2	4	2	6	8	6¼	58	4	58	6	8
½	4	8	4	13	4	6½	60	8	60	13	4
¾	7	0	7	0	0	6¾	63	0	63	0	0
1	9	4	9	6	8	7	65	4	65	6	8
1¼	11	8	11	13	4	7¼	67	8	67	13	4
1½	14	0	14	0	0	7½	70	0	70	0	0
1¾	16	4	16	6	8	7¾	72	4	72	6	8
2	18	8	18	13	4	8	74	8	74	13	4
2¼	21	0	21	0	0	8¼	77	0	77	0	0
2½	23	4	23	6	8	8½	79	4	79	6	8
2¾	25	8	25	13	4	8¾	81	8	81	13	4
3	28	0	28	0	0	9	84	0	84	0	0
3¼	30	4	30	6	8	9¼	86	4	86	6	8
3½	32	8	32	13	4	9½	88	8	88	13	4
3¾	35	0	35	0	0	9¾	91	0	91	0	0
4	37	4	37	6	8	10	93	4	93	6	8
4¼	39	8	39	13	4	10¼	95	8	95	13	4
4½	42	0	42	0	0	10½	98	0	98	0	0
4¾	44	4	44	6	8	10¾	100	4	100	6	8
5	46	8	46	13	4	11	102	8	102	13	4
5¼	49	0	49	0	0	11¼	105	0	105	0	0
5½	51	4	51	6	8	11½	107	4	107	6	8
5¾	53	8	53	13	4	11¾	109	8	109	13	4
6	56	0	56	0	0	12	112	0	112	0	0

COMPOSITIONS OF SOLDERS

Metal	Tin	Copper	Zinc	Antimony	Lead	Silver	Gold	Brass
For lead	1	—	—	—	1½	—	—	—
,, tin	1	—	—	—	2	—	—	—
,, pewter . . .	2	—	—	—	1	—	—	—
,, brazing (hardest) .	—	3	1	—	—	—	—	—
,, brazing (hard) . .	—	1	1	—	—	—	—	—
,, brazing (soft) . .	1	4	3	—	—	—	—	—
,, brazing (soft) . *or*	2	—	—	1	—	—	—	—
,, iron	—	2	1	—	—	—	—	—
,, steel	—	3	1	—	—	19	—	—
,, silver (hard) . .	—	1	—	—	—	4	—	—
,, silver (soft) . .	—	—	—	—	—	2	—	1
,, gold	—	1	—	—	—	2	24	—

FLUXES FOR SOLDERING OR WELDING

Metal	Flux
Iron or steel . .	Borax and sal-ammoniac
Tinned iron . .	Resin or chloride of zinc
Copper and brass .	Sal-ammoniac or chloride of zinc
Zinc . . .	Chloride of zinc
Lead . . .	Tallow or resin
Lead and tin pipes .	Resin and sweet oil

Concrete

Concrete is made in three classes according to the percentage of cement used.

1. Strong, containing about 15 per cent (say a mixture of 1 cement : 2 sand : 4 aggregate) used for floors, culverts, arches, retaining walls, walls for dust chambers, and for foundations in wet places. The above is the one mostly used.

2. Medium, containing about 10 per cent cement (say 1 : 3 : 6) used for foundations of machinery and furnaces, ground floors, bottoms of flues, dust chambers, etc.

3. Poor, containing 7 to 8 per cent (say 1 : 4 : 8) used for levelling the bottom of excavations preparatory to commencing a foundation proper, and for foundations not called on to support a weight over 6 t/sq ft.

The aggregate may be rounded pebbles or broken stone. The former offers less surface to be covered weight for weight of the same aggregate, but the pieces do not mesh in so well as angular pieces. The larger aggregate generally passes through a 2 in. ring and gives more strength to the concrete than smaller stones.

Sand is used to fill voids in the aggregate: it should be free from organic matter (animal or vegetable). It is better to have, say, 15 per cent finer material with the sand than to have it all the same size, since it helps to fill in the voids better.

Cement is used to coat all surfaces so as to bind the particles together and also fill up voids in the sand.

To ascertain voids in the aggregate, fill a box of one cubic foot capacity with the aggregate, then measure the quantity of water it takes to fill the box. That is the quantity of sand required. Then take the amount of sand required, place it in a box and measure the quantity of water necessary to saturate it. This gives the amount of cement required. Suppose the voids in the aggregate require to be filled with sand to be 33 per cent, then the proportion of sand to the aggregate will be in the proportion of 1 : 3. If the voids in the sand also came to 33 per cent, the proportion of cement to sand would also be 1 : 3, and the theoretical proportion of all these would be 1 : 3 : 9; but as it is desirable to make allowance for unequal mixing, the amount of cement should be increased by say 10 per cent. Eleven bags of cement go to the ton.

Small quantities of concrete are mixed by hand; larger quantities in batch or continuous machines. When mixing by hand, first mix sand and cement dry, then add the wet aggregate and mix again: the whole is then sprinkled with water and turned repeatedly. The more the concrete is turned, the better the result. In batch machines, the sand and cement are thoroughly mixed dry; then mixed wet and finally the aggregate is added and mixed. If concrete is made too dry, it shows imperfections on the surface. If made too wet, it shows a good surface but is botched inside. It should be wet enough to make the mass quail under hard ramming which flushes the surface, but no wetter. All concrete must be well rammed as this greatly increases its strength.

Fresh concrete must be protected from the weather till it has set. If the weather is hot and the moisture likely to evaporate, it should be covered with wet sacks. Frost damages concrete before setting, as when the water freezes it expands; it also delays hardening. When placing concrete in two or more layers, it is desirable to place the top layer before the first has set. Where this is not possible, the first layer should be scratched before it sets.

Concrete floors are made of a 1 : 3 : 5 mixture and on this is placed a finishing coat of half to an inch thick according to the amount of use the floor is likely to be put. The coat is made in the proportion of 1 cement to ½ sand, or 1 cement to 2 sand: that with the larger amount of sand standing the wear of traffic better, and that richer in cement being more impervious to water. An addition of bentonite to the mixture helps to make it waterproof. Expansion joints must be left if considerable changes in temperature are expected, or steel mesh can be used. Such a floor is not ready to be used for at least a week after completion. The upper surface must be kept damp while setting.

Brick Stack or Chimney

A brick stack may be square, octagonal or round in cross-section. Being heavy it must have a solid foundation. The pedestal is generally of concrete laid down in one foot courses. The height of the pedestal varies between one-tenth to one-twelfth the total height of the shaft, and its area should be one-sixth the height of the stack. The pedestal should be allowed to stand some time before erecting the superstructure. A door is placed at the bottom to allow the flue dust to be cleaned out. The brickwork should not be built more than two feet to two and a half feet high at a time so as to give the mortar time to set. The bricks should be selected, hard, sound, with sharp edges. A suitable bond is two to four courses of stretchers to one of headers. Build in iron hoops edgeways every few courses to strengthen the stack. Give the stack a batter of 2½ in. for every 10 ft in height. The brickwork at the top of the stack should be at least 9 in. for 20 ft and increase 4 in. for each 20 ft lower down. Any set off must be on the outside of the stack, as the inside diameter must remain the same throughout. Any fire brick lining should be independent of the shell, an air space of 4 in. being left between them. This space is closed at the top, but the last two or three courses are built open. A coping stone is placed on the top of the stack. A lightning conductor is placed at the top of the stack connected by copper tape to an iron plate embedded in moist ground.

There are three methods by which a brick stack may be demolished, depending on whether there is room for the stack to fall, or whether there are buildings in the neighbourhood that might be damaged.

1. In the latter case the stack must be taken down brick by brick and sent below through an air-tight wooden pipe with an air-tight door at the bottom. The pipe is made a little larger than the width and breadth of a brick to allow for any mortar clinging to it and also to permit the escape of air (say ¼ in. on each side). The bricks are sent down end on, and being cushioned by the air they reach the bottom unbroken. The wooden pipe is shortened as the height of the stack is reduced. When the pipe is full of bricks, the air-tight door is opened, and the bricks removed.

2. Several holes are drilled in the base of the stack, charged with blasting powder, and all fired simultaneously by electricity.

3. Cut into one side and prop up with wood. Fasten chains or ropes to the props and pull them out. If the stack does not fall, fasten a rope round the stack and pull it down with horses, bullocks or an engine.

Straightening a Stack. Sometimes a stack or similar structure gets out of plumb owing to a part of the foundation giving way, or perhaps it is due to a leakage of water softening the ground. This may be rectified by digging out earth under the high side and allowing the stack to settle, but this is dangerous. Another method, worse than the first is sometimes employed, when some bricks are gradually taken out on the high side and replaced by narrower bricks. A better way is to dig a trench of sufficient length, width, and depth, according to the nature of the work to be carried out, along the higher side of the structure then to work two-inch diameter steel pipes under the stack half way, the pipes being 12 in. apart. Withdraw the pipes and clean the earth out of them: they are again worked under the stack half way, between the first holes and again withdrawn. In this way the weight of the stack settles down gradually by crushing the earth between the holes. Of course the amount the stack is out of plumb has to be determined beforehand, and plumb lines are used to be seen if the stack is settling properly.

Concrete Stacks

Stacks are also formed of reinforced concrete, but on account of the thinness of the walls no stones are included, the mixture consisting of 1 part of Portland cement to 3 of sand. The work of concreting is carried on from inside the stack, the men standing on a platform erected on scaffolding. The scaffolding is built up a few feet in advance of the concrete. From this project beams from which the inner and outer moulds are suspended by means of vertical rods. The moulds of the last section concreted are held in position by friction against the concrete: they are removed and placed in position for the next section. The vertical steel bars for reinforcement are held in proper alignment by a circular wooden gaugè six feet above the top of the mould being filled and can be raised as work proceeds: rings of $\frac{5}{8}$ in. steel are wired to the vertical rods, every 18 in. The moulds are made in sections 3 ft high which when coupled together form a complete ring. There is a double shell, the outer may be from 12 to 9 in. thick, depending on the height of the stack, but becomes thinner as it approaches the top. The inner shell may be 4 or 5 in. thick, and is separated from the outer by an air space of 4 in. This space is connected with the atmosphere by portholes through the bottom of the outer shell. The desirable features in a concrete stack are the smoothness of the inner surface, low conductivity of heat, and ability to resist high temperatures. Stacks are given a batter of one inch to eight feet.

Steel Stacks

If a good rock foundation cannot be had steel stacks are better than those of brick or concrete as they are lighter. The base of a steel stack forms the frustum of a cone for some 18 to 20 ft, the bottom of which is fitted with a ring of very strong angle-iron, through which are holes for holding-down bolts fixed to the foundation. The upper portion is riveted to the chimney plates.

There are two methods used for erecting steel stacks. If there is sufficient ground room, the sections can be riveted together on the ground and raised in position in one piece if short, or in lengths if tall. The lengths of eight or ten rings are hoisted in position by blocks and tackle attached to the top of two poles placed opposite each other on either side of where the stack is to be. At first the poles are fixed in the ground, but for higher lengths they are attached to that portion of the stack already erected. The different lengths have flanges attached top and bottom to enable them to be bolted to the sections below and above.

If the different rings cannot be riveted together on the ground, they are built up on the stack itself. There are two platforms, one inside and one outside. The inside one is circular, a little smaller than the inside of the stack. In the centre of it is a footing for a tube, into which the upright of a small crane fits: there is also a manhole. This platform is raised or lowered by means of four ropes which pass through pulleys. The lower pulleys are hooked on to eyebolts attached to the platform, the upper pulleys hang on hooks which pass through rivet holes at the top of the highest fixed section. When a section has been riveted in position, the platform is hauled up as far as possible, and the hooks transferred one by one to the next row of rivet holes. The outer platform consists of a framework of two angle-irons, bent at right

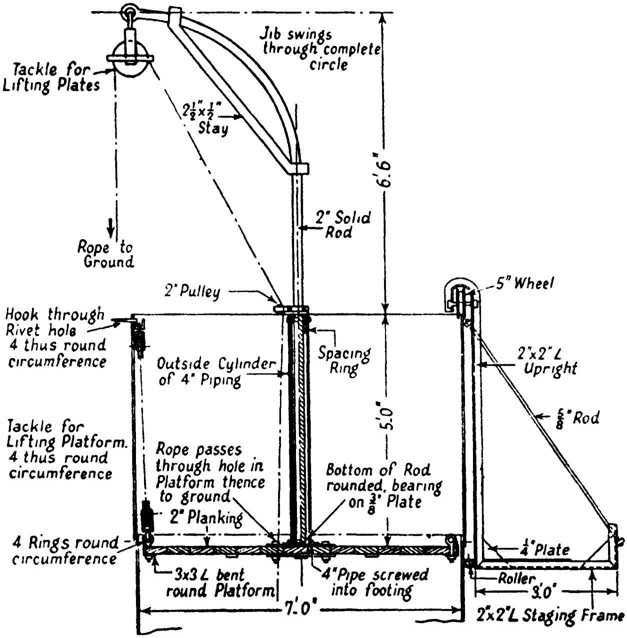

Jib swings through complete circle
Tackle for Lifting Plates
2½"x½" Stay
6'.6"
2" Solid Rod
Rope to Ground
2" Pulley
5" Wheel
Hook through Rivet hole 4 thus round circumference
Outside Cylinder of 4" Piping
Spacing Ring
2"x2" L Upright
5'.0"
⅝" Rod
Tackle for Lifting Platform. 4 thus round circumference
Rope passes through hole in Platform thence to ground
Bottom of Rod rounded, bearing on ⅜" Plate
2" Planking
4 Rings round circumference
¼" Plate
3x3 L bent round Platform
4" Pipe screwed into footing
Roller
3.0'
7'.0"
2"x2" L Staging Frame

FIG. 77. STEEL STACK ERECTION

angles, the free ends being connected with an iron rod. Planks connect the two angle-irons, to which they are bolted, for the outside riveter to stand on. The top of each upright of this platform is bent over to form a hook, and a hole is drilled through the hook for the axle of a grooved 5 in. wheel. The wheels can then run round the edge of the last erected section and at the same time support the platform. Two small horizontal wheels are connected to the angles of the framework to help the platform roll against the structure when moving it. For a 180 ft stack, the lowest 60 ft section can be ⅜ in. plate; the next 60 ft $\frac{5}{16}$ in. and the top section ¼ in. thick.

Belting

Belts may be of leather, or canvas and rubber. Leather belts may be single, double, or triple in thickness, the average working tensions of which in practice are 40, 70, and 100 lb respectively. Leather belts should be used with the flesh or stronger side on the outside, and the grain or hair side in the inside next the pulley as it adheres better. The motion of the drive should be with the laps of the belt, not against them. The direction of the laps where the strips of leather have been connected can be felt by running a finger backward and forward over the flesh side.

The ends of a belt must meet squarely. When punching holes for lacing, use oval punches, the longer diameter being parallel to the length of the belting. The holes should not be nearer the ends of the belt than ⅞ in., or nearer the sides than ¾ in.; on wide belts these distances should be greater. Light belts have one row of holes at each end; heavy belts have two rows at each end. When two rows are used, the lacing should be arranged zigzag. In a 6 in. belt there should be 7 holes, 4 of which are in the row nearer the end. A 10 in. belt has 9 holes, 5 of which are in the row nearest the end. The lacing thongs are of leather or raw hide. The tension of each lacing must be uniform. The straight part of the lacing is on the pulley side and the cross part on the outside. The lacing thongs vary in thickness from ¼ in. to ⅜ in., and from $\frac{1}{64}$ to nearly ⅜ in. in width. They should just fill the punched holes. When finished lacing, nick the ends of the lace like the barbs of an arrow to prevent it coming undone. If metallic fasteners are used, they should be of some soft metal that will not scratch the surface of the pulleys.

To decide the length of a belt to pass over two pulleys, the easiest way is to stretch a steel tape over the pulleys in the same position that the belt will have, but if this is not convenient, add the diameters of the two pulleys together, divide by two, multiply the quotient by 3·25, and to the product add twice the distance between the centres of the shafts. A new belt will stretch at first and will have to be cut and rejoined, but the belt must not be too tight.

There is always a certain amount of slippage. Resistance to slipping depends on the coefficient of friction, the area of surface in contact between belt and pulleys, and the pressure on the pulley face due to belt pull. The coefficient of friction depends on the materials in contact. For ordinary iron or wooden pulleys and leather belting the coefficient is 0·3 to 0·4 for dry surfaces, and, say, 0·15 to 0·2 for wet and oily surfaces. The friction of leather on leather is five times that of leather on iron.

Belting should be inspected periodically, kept clean, and dressed with reliable dressing to keep it pliable, such as tanner's dubbin, used with judgment. Before dressing, a belt should be washed with lukewarm water and when dried have the grease well rubbed into it. Belts should be kept free from lubricating oil. Belts should be cleaned and dressed without taking them off their pulleys, otherwise they will shrink.

Should the laps of a leather belt start, they may be fastened down with the following cement. Equal proportions of glue and Prussian gelatine dissolved in water and cooked in a tin vessel, set in a larger one containing water, and raised off the bottom. Cook till quite thick and ropy. Work this into places where the laps have started with a knife, and hammer till dry. A few brass sprigs may be used; they are cut off a little from the surface on the reversed side and hammered down on a flat iron.

Another cement for leather belting is composed of two parts by weight of pitch, four parts of india-rubber, one part of shellac, four parts linseed oil, and ten parts gutta-percha. The solid materials are cut up into small pieces, melted, mixed over a slow fire, the linseed oil added and applied while hot.

The best drive for horizontal open belts is with the tight side below, since the sag of the belt on the upper side increases the angle of wrap on the power pulley. With vertically arranged belts, the lower pulley being the driver, any stretch in the belt causes considerable slipping which means wear on the belt, so an idler pulley is required to maintain the necessary tension. When putting on a belt, it should be placed on the driving pulley last. When a belt is slipped off a pulley, do not let it chafe against the shafting, but tie it up, or place a saddle of wood between it and the shafting for the belt to rest on.

When possible, arrange for power to be taken off from both sides of the main shafting so as to equalize the strain on it. Cross belts used for reversing the rotation of a counter shaft should be avoided when possible, as the belt rubs at the point of crossing which increases its wear.

Rubber belting is made of several thicknesses or plies of canvas covered with rubber and pressed together. It can be used in wet places and where drives are subject to acid fumes. The quality of the rubber determines the grade of the belt; the strength depends on the weight of the canvas used which is rated in ounces per square yard. The safe working strain for a rubber belt 1 in. wide varies from 15 lb per ply for a 3-ply belt, up to 18 lb per ply for an 8-ply belt. A rubber belt can be made in one piece instead of in strips as in the case of leather belts.

Any increase of speed over 3000 to 4000 ft/min decreases the power transmitted as the centrifugal force tends to throw the belt away from the pulley.

When rope drives are used, the rope should be cable laid as it makes a smoother splice; and the rope must be run in iron grooves, not wooden ones.

Knots, Hitches, and Lashings

Different knots are tied for a definite purpose, so it is necessary to know the requirements beforehand.

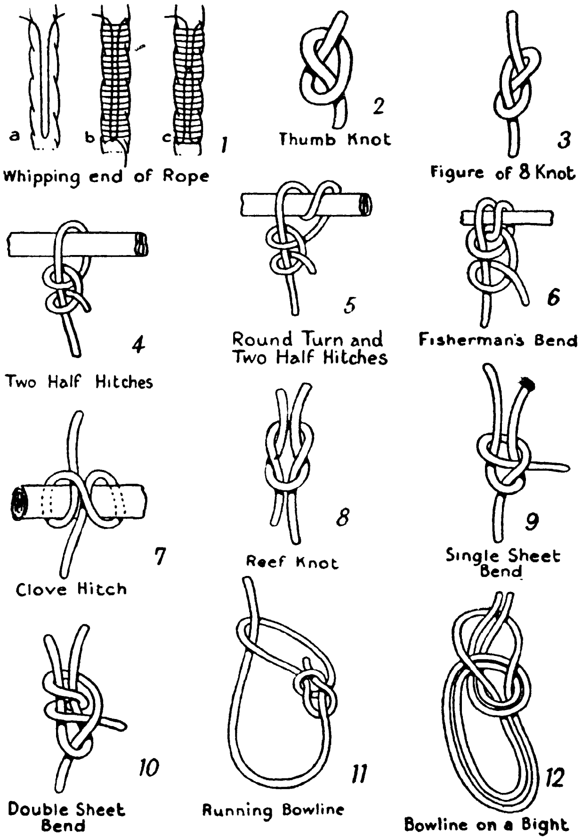
Whipping. (1) This is tying a piece of twine round the end of a rope to prevent the strands from untwisting. There are several ways of doing this. One method is to lay a loop of the twine along the rope near its end; the longer end of the twine is then wound round and round both the rope and the loop a few times; the end of the twine is then passed through the loop, and the other end pulled till the loop is in the middle of the whipping, when the ends are neatly cut off.

Simple Overhand or Thumb Knot. (2) Used as a stop on a rope to prevent the end from slipping through a block.

Figure of Eight Knot. (3) Also used as a stop on a rope. It is less injurious to the fibres of a rope than a thumb knot, and is easier to undo.

Two Half Hitches. (4) Used for securing the loose ends of lashings; also for securing a rope to timber when no great pull is expected.

Round Turn and Two Half Hitches. (5) For belaying or



1 Whipping end of Rope

2 Thumb Knot

3 Figure of 8 Knot

4 Two Half Hitches

5 Round Turn and Two Half Hitches

6 Fisherman's Bend

7 Clove Hitch

8 Reef Knot

9 Single Sheet Bend

10 Double Sheet Bend

11 Running Bowline

12 Bowline on a Bight

Fig. 78a. Knots, Hitches, and Lashings

making a rope fast to an anchorage, so that the strain on the rope shall not jam the hitch.

Fisherman's Bend. (6) For bending a rope to a small anchor; or attaching the end of a rope to the bail of a bucket.

Clove Hitch. (7) Used for commencing and finishing a lashing; for fastening guy-ropes to the head of a derrick; it will take the strain at either end without slackening.

Reef Knot. (8) This must not be confused with a granny-knot which will slip. The reef knot is used for tying two ropes of the same size together. The two short ends and two long ends should be on the same side respectively. Used for reefing sails; for tying parcels, and the ends of bandages.

Single Sheet Bend. (9) Used for tying two ropes of unequal sizes together.

Double Sheet Bend. (10) Used for greater security under the same circumstances as the single sheet bend. The second turn round the bight of the rope prevents jamming. Also used for connecting wet ropes of different sizes.

Bowline. (11) This is a loop made at the end of a rope that will not slip. It is used for mooring boats; for fastening a rope to a bucket. A man being lowered down a shallow shaft can place a foot in it while guiding himself with his other foot.

Bowline on a Bight. (12) This consists of two loops which can be made on any part of a rope. It can be used for a man to place a leg in each loop in a sitting position when being lowered or raised in a shallow shaft; or one loop may be placed under his armpits and the other under his thighs. As a man harness, one loop is placed over each shoulder.

Timber Hitch. (13) For fastening a rope to a windlass; for starting lashings on scaffoldings.

Killick Hitch. (14) The same as a timber hitch with the addition of a half hitch a little farther away. Used for dragging, lowering, or hoisting long objects, such as timber and pipes.

Rolling Hitch. (15) Used for attaching a rope to a spar when it is required to raise it into position, as in the case of scaffolding or telegraph poles. Used because it will not slip.

Sheep Shank. (16) For shortening a rope temporarily and quickly.

Wool Hitch. (17) For tightening a rope thrown over a loaded wagon; for tightening guy-ropes. It serves the purpose of a temporary block and tackle.

Stopper Hitch. (18) Used when necessary to take the strain off a rope temporarily which is already stretched; such as

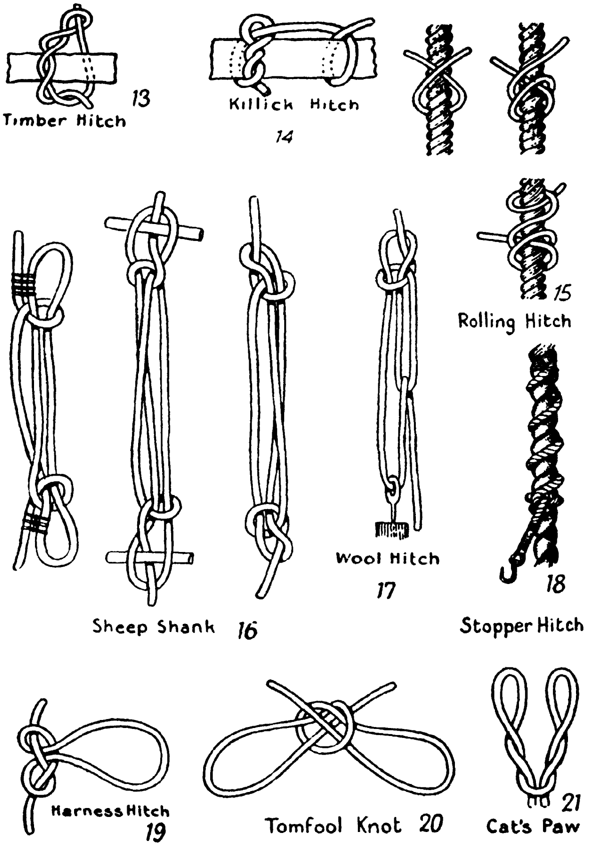


FIG. 78B. KNOTS, HITCHES, AND LASHINGS (*cont.*)

when testing the safety catches of a cage from the drum of the engine, if a suitable clamp is not available.

Harness Knot, or Hitch. (19) This is a loop made on a rope generally placed over the shoulder of a man to enable him to draw an object. Several such knots can be made on a long rope.

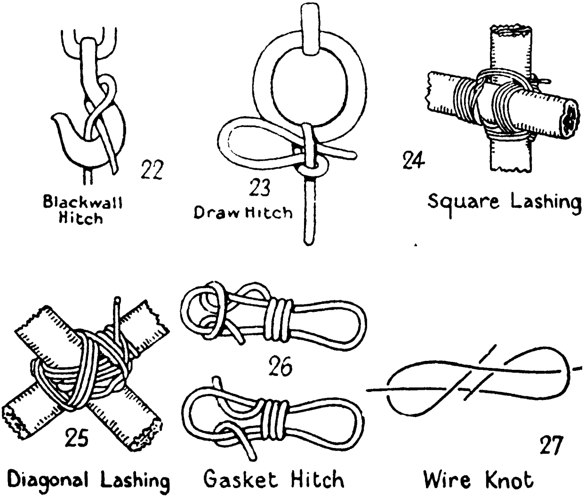


FIG. 78C. KNOTS, HITCHES, AND LASHINGS (*cont.*)

Tomfool Knot. (20) Two loops are tied in a thumb knot. It can be used to manacle a man's hands behind his back. A hand is placed in each loop which are then drawn tight and frapping turns taken between the hands to tighten the loops still more; the ends of the rope are then tied in a reef knot. This is the first stage of the Chair or Fireman's Knot, used for raising or lowering a man in a shaft, or lowering from a burning building. A struggling or insensible person cannot fall out of it. Useful when sampling the sides of a shaft when both hands are required to be free. Make one loop out of a length of rope as far as one can stretch from one hand to the other. Make another loop out of a length of rope as far as one can stretch from one extended arm to the shoulder of the other arm, and tie these together into a tomfool knot. The loops are now adjusted if necessary: place a half hitch over each loop to prevent them from slipping. The legs are passed through the larger loop which is placed under the thighs. The smaller loop is placed over the head and under the armpits. The knot should be in front of the person in the sling.

Cat's Paw. (21) For securing a sling to a crane hook.

Blackwall Hitch. (22) Used with a pliant rope. Convenient for returning an empty rope to a hoist. Used to secure a rope to the hook of a block.

Draw Hitch. (23) This will stand a give and take motion, such as that of a boat afloat moving up and down by waves, or of a tied-up horse. The object is easily set free in case of an emergency.

Square Lashing. (24) For lashing two spars together which are at right angles to one another.

Diagonal Lashing. (25) For lashing two spars together which are not at right angles to one another.

Gasket Hitch. (26) When manilla ropes are not in use, they should be coiled clockwise, fastened with a Gasket Hitch and hung up in a dry place.

Wire Knot. (27) Used for connecting fencing wires.

Round Ropes

Let W = Breaking load in tons
C = Circumference of rope in inches.

Hemp ropes $W = 0{\cdot}25\ C^2 \therefore C = \sqrt{\frac{W}{0{\cdot}25}}$

Iron wire ropes $W = 1{\cdot}50\ C^2 \therefore C = \sqrt{\frac{W}{1{\cdot}5}}$

Crucible steel wire ropes $W = 3\ C^2 = \sqrt{\frac{W}{3}}$

Improved plough steel wire ropes $W = 4\ C^2 \therefore C = \sqrt{\frac{W}{4}}$

Let A = Area in square inches
d = Diameter in inches.

$$C = 3{\cdot}1416\ d = \sqrt{12{\cdot}5664\ A}$$

$$A = 0{\cdot}7854\ d^2 = \frac{Cd}{4} = 0{\cdot}07958\ C^2$$

$$d = \frac{C}{3{\cdot}1416} = 0{\cdot}3183\ C = \sqrt{\frac{A}{0{\cdot}7854}}$$

Let L = Load in tons
M = Factor of safety (from 6–10)
F = Depth of shaft in fathoms
w = Weight (roughly) in lb/fm

For hemp—

$$C = \sqrt{\left\{\frac{L}{\frac{0\cdot25}{M} - \frac{F}{4 \times 2240}}\right\}}$$

For iron—

$$C = \sqrt{\left\{\frac{L}{\frac{1\cdot5}{M} - \frac{F}{1\cdot2 \times 2240}}\right\}}$$

For crucible steel—

$$C = \sqrt{\left\{\frac{L}{\frac{3}{M} - \frac{F}{1\cdot2 \times 2240}}\right\}}$$

For improved plough steel—

$$C = \sqrt{\left\{\frac{L}{\frac{4}{M} - \frac{F}{1\cdot2 \times 2240}}\right\}}$$

Round Taper Ropes

Let A = Area of rope at any point in square inches
a = Area of rope at bottom end in square inches
w = Weight of one cubic inch of the rope in lb. (Approx. for iron and steel, W = 0·14; for hemp rope, W = 0·043)
L = Safe load in lb/sq in. of section of rope. (Say, iron, 7000; steel, 11,500; plough steel, 13,440; hemp, 740)
D = Distance in inches from A to a
W = Weight of rope in lb
e = 2·7182.

$$A = ae^{\frac{wD}{L}}$$

$$W = La\left(e^{\frac{wD}{L}} - 1\right) = L(A - a)$$

Flat Ropes

The strength of flat ropes is equal to the sum of the strength of the round ropes of which it is made, minus 10 per cent.

WEIGHTS AND BREAKING LOADS OF MANILLA ROPES

Diameter	Circumference	Weight per Foot	Breaking Load	
In.	In.	Lb	Tons	Lb
0·239	¾	0·019	0·25	560
0·318	1	0·033	0·35	784
0·477	1½	0·074	0·70	1,568
0·636	2	0·132	1·21	2,733
0·795	2½	0·206	1·91	4,278
0·955	3	0·297	2·73	6,115
1·11	3½	0·404	3·81	8,534
1·27	4	0·528	5·16	11,558
1·43	4½	0·668	6·60	14,784
1·59	5	0·825	8·20	18,368
1·75	5½	0·998	9·80	21,952
1·91	6	1·19	11·4	25,536
2·07	6½	1·39	13·0	29,120
2·23	7	1·62	14·6	32,704
2·39	7½	1·86	16·2	36,288
2·55	8	2·11	17·8	39,872
2·86	9	2·67	21·0	47,040
3·18	10	3·30	24·2	54,208
3·50	11	3·99	27·4	61,376
3·82	12	4·75	30·6	68,544
4·14	13	5·58	33·8	75,712
4·45	14	6·47	37·0	82,880

The above figures are for ropes of average quality, as pieces from the same coil may vary 25 per cent.

A few months of exposed work weakens ropes 20 per cent to 50 per cent.

EXAMPLE. What is the weight of 300 ft of a 2·07 in. diameter manilla rope?

Opposite 2·07 in., under the heading of "weight per foot," is found 1·39 lb; this multiplied by 300 will give the desired weight, 1·39 lb × 300 ft = 417 lb.

A rope wound round a barrel offers the following frictional resistance to sliding—

24 when the rope is wound 1 time round the barrel
111 ,, ,, 1½ ,, ,,
535 ,, ,, 2 ,, ,,
2575 ,, ,, 2½ ,, ,,

Wet ropes expand in diameter and contract in length.

Tarred ropes are weaker than white ropes.

When measuring the diameter of a rope, measure from the outside of the strands, not from the spaces in between. It is better to measure the circumference with paper or string.

The factor of safety for a steel wire rope when slow winding, is generally taken at one-sixth the breaking strain.

When splicing manilla ropes, allow the following lengths over and above the net weight: 1 in. diam., 9 ft; 1¼ in. diam., 10 ft; 1½ in. diam., 12 ft; 1¾ in. diam., 13 ft; 2 in. diam., 14 ft.

Deterioration of Ropes

The occasional breakage of a wire in a new rope need not cause alarm, as it is due to the fact that now and again the junction where two wires have been brazed happen to come on the crown of the rope where there is most wear. The broken wire soon takes up the strain again. A rope may deteriorate owing to corrosion or abrasion. Galvanizing reduces the strength of steel some 20 per cent. A wire rope should not be coiled or uncoiled like a hempen rope, but when not on a reel should be rolled on the ground to prevent kinking or untwisting. New ropes generally gain 2 to 3 per cent in length when strain is put on them.

The parts of a hoisting rope where deterioration is most likely to occur are at the shackle, which is subject to violent jerks on starting to hoist; where the rope rests on the pulley at the end of an upward or downward journey, as the strands are liable to open and allow moisture to enter; also where the rope is partly sheltered in the engine house and partly exposed to the weather. The rope which passes under the drum wears out more quickly than the other on account of the reverse bending. A rope may be worn out by bad driving if the driver starts with a jerk, or hoists too fast when the load is near the surface and has to put on the brakes excessively, causing the rope to grind against the pulley. The bright appearance of a

(*a*)
Wire rope, showing good wear

(*b*)
Wire rope, showing strands broken from undue bending

(*c*)
Wire rope, showing effect of abrasion

FIG. 79. WIRE ROPES

rope is no proof that the inside wires are not rusted. Internal wear of a rope is caused by friction of the wires on each other and the fibres of the core.

No hoisting rope should be used for more than three years, no matter how good it may appear to be.

There is no standard rule when to discard a hoisting rope. A rope discarded for hoisting men may still be used for hoisting ore for a longer period. The various rules are based on—

(*a*) The strength of the rope.
(*b*) The number of broken wires in a given length of rope.
(*c*) The amount of wear of the wires at the crown of the strands.
(*d*) The length of time the rope has been in service.
(*e*) The number of trips it has made.
(*f*) The ton-mileage hoisted by the rope.

A rope should be cleaned and dressed periodically. Rope dressing consists of graphite mixed with vaseline, linseed, palm or other vegetable oil, and applied hot.

Safety Precautions. Before any rope or chain is used for hoisting or hauling purposes, it should be tested to carry twice the ordinary working load.

A hoisting rope should be examined at least once a month by passing it through a gloved hand or a handful of cotton waste to feel for broken wires, at a rate not exceeding 50 ft a minute. At the same time a close look-out should be maintained for minor defects, such as excessive wear, loosening of the strands, kinking, corrosion, etc. When the rope shows signs of wear it should be examined more frequently, once a fortnight or even once a week. A few feet should be cut off the end of the rope nearest the cage every three months if in constant use, or every six months if only used intermittently. The strands should be untwisted and the wires tested separately for tension and flexibility, and the inner wires examined for corrosion. The wires of a new rope will stand from 14 to 17 flexions through an angle of 180°, depending on the diameter of the wire, when held in a vice with jaws rounded to $\frac{3}{16}$ in. The factor of safety should always be from one-fifth to one-eighth the breaking strength of the rope to allow for wear, corrosion, severe strain due to starting, etc. Ropes used for lowering men must be in one piece, not spliced.

Bridle Chains. Bridle chains tend to preserve winding ropes from sudden jars. They become brittle in time owing to careless winding. They should be carefully cleaned and examined at least every six months, and annealed in a furnace

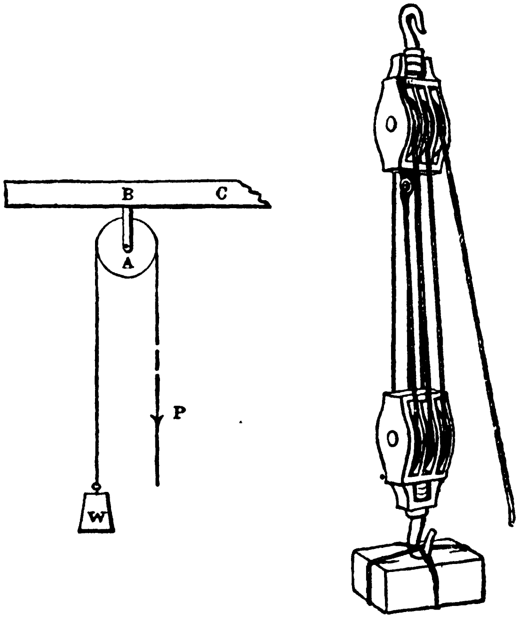


Fig. 80. Raising a Weight
(*a*) Single fixed pulley; (*b*) Block and tackle

at an even temperature of about 1560° F. at least once a year.

Safety-detaching Hooks. An approved safety-detaching hook should be used between the hoisting rope and bridle chains of every cage or skip. It should be kept clean and well lubricated. It should be taken to pieces at least once a month to be examined, cleaned, oiled, and put together again by a competent person. It should be annealed periodically to overcome crystallization. There should be at least

10 ft clearance between the detaching hook and point of detachment when the cage is at the landing. There should be automatic keeps near the top of the headframe for the cage to rest on in case of an overwind.

Block and Tackle

With a single fixed pulley there is no gain of power or speed in raising a weight, but if the weight is attached to a movable pulley, the rope has one end fixed, then passes under the lower pulley and over another above, the force required to balance the weight is halved; but the time required is doubled since there is more rope to pull on. With an ordinary block and tackle, the fixed end of the rope is attached to a hook below the upper block, the rope is then passed round each of the sheaves of the lower and upper blocks in succession until it terminates where the power is applied. As there is only one rope, the strain is the same all along the rope. The mechanical advantage of double sheave blocks (leaving the weight of the lower block and friction out of the question) is equal to the number of rope lengths supporting the lower block. Thus we get the rule that, "In any combination of pulleys where one continuous rope is used, a load on the free end will balance a weight on the movable block as many times as great as the load on the free end as there are parts of the rope supporting the load, not counting the free end."

Chains

Let W = Breaking load in tons
D = Diameter in sixteenths of an inch

$$W = \frac{D^2}{9};\ D = \sqrt{9W}$$

The greatest working load for studded link cables is one-half greater than that for close-link crane chains of the same diameter of iron.

The strength of chains varies as the square of the diameter of the iron in the link.

Chains are generally proved to about half their ultimate tensile strength or double the intended load. The link is made as short as possible consistent with easy play, so that they shall not bend on a pulley. They should be annealed by making them red hot and then cooling slowly in the air. Any defect in the welding will be noticed while the links are hot.

STRENGTH AND WEIGHT OF CLOSE-LINK CRANE CHAINS AND SIZE OF EQUIVALENT HEMP ROPE

Diameter of Iron	Weight of Chain	Breaking Strength	Testing Load	Girth of Equivalent Rope	Weight of Rope	Greatest Working Load
In.	Lb/fm	Tons	Tons	In.	Lb/fm	
$\frac{1}{4}$	3·5	1·9	0·75	2	$1\frac{3}{8}$	One-half the test load
$\frac{5}{16}$	6·0	3·0	1·10	$2\frac{1}{2}$	$1\frac{1}{2}$	
$\frac{3}{8}$	8·5	4·3	1·6	$3\frac{1}{4}$	$2\frac{1}{2}$	
$\frac{7}{16}$	11·0	5·9	2·3	4	$3\frac{3}{4}$	
$\frac{1}{2}$	14·0	7·7	3·0	$4\frac{3}{4}$	5	
$\frac{9}{16}$	18·0	9·7	3·8	$5\frac{1}{2}$	7	
$\frac{5}{8}$	24·0	12·0	4·6	$6\frac{1}{4}$	$8\frac{1}{2}$	
$\frac{11}{16}$	28·0	14·6	5·6	7	$10\frac{1}{2}$	
$\frac{3}{4}$	31·5	17·3	6·8	$7\frac{1}{2}$	12	
$\frac{13}{16}$	37·0	20·4	7·9	$8\frac{1}{4}$	15	
$\frac{7}{8}$	44·0	23·1	9·1	9	$17\frac{1}{2}$	
$\frac{15}{16}$	50·0	26·1	10·5	$9\frac{1}{2}$	$19\frac{1}{2}$	
1	56·0	29·3	12·0	10	22	
$1\frac{1}{8}$	71·0	36·3	15·3	$11\frac{1}{4}$	$27\frac{3}{4}$	
$1\frac{1}{4}$	87·5	44·1	18·8	$12\frac{1}{2}$	$34\frac{1}{2}$	
$1\frac{3}{8}$	105·8	52·8	22·6	$13\frac{3}{4}$	$41\frac{1}{2}$	
$1\frac{1}{2}$	126·0	62·3	27·0	15	$49\frac{1}{2}$	

Power

A man-power is equal to the raising of 50 lb 1 ft high per second. Nominal horse-power is a commercial term which does not actually represent the horse-power of an engine.

$$\text{N.H.P.} = \frac{\text{diam. of cylinder}}{30}$$

The indicated horse-power is that developed in the cylinder and shown on the diagram when an engine is indicated.

An English horse-power is 33,000 ft lb/min = 42·75 British thermal units per min.

A metric horse-power can raise a kilo one metre in 75 sec = 1·0139 English horse-power.

For the actual horse-power exerted by an engine—

Let p = the mean effective pressure of steam in lb/sq in. less 3 lb/sq in. frictional allowance

A = area of the cylinder in square inches

L = length of the stroke in feet
N = the number of strokes per min. = revolutions × 2
H.P. = the horse-power.

Then—

$$\text{H.P.} = \frac{pLAN}{33{,}000}$$

A horse can draw up a 30 gal bucket 450 ft in 5 min by means of a whim.

A draught horse is most efficient when walking at a slow speed (2 to 3 m.p.h.), and can draw 22,000 lb/min or about ⅔ of a mechanical horse-power. On steep grades or on heavy patches, extra horses should be used. If two or more teams are travelling together, this may be done by taking the horses from one team to assist another. Where there are sharp turns on a road it is better to use horses than bullocks.

A horse put down a pit at about 5 to 7 years should travel from 14 to 16 miles a day on a fairly level track at a walking pace. A pony should not be put into a pit under 3 years. A pony is less than 13 hands high: a hand is 4 in. A horse can carry about 27·5 times its own weight on its back.

A boiler horse-power is 30 lb water evaporated into steam of 70 lb pressure.

Theoretical horse-power for water—

$$\frac{\text{Weight of water in lb} \times \text{effective head in feet}}{33{,}000}$$

Steam Boilers

The boiling point of any liquid is that point on the temperature scale when the tension throughout its mass just overcomes the surrounding pressure.

The boiling points of fresh water at different pressures are approximately as follows—

Under a	pressure of	⅛	atmosphere	123° F.
,,	,,	¼	,,	150° F.
,,	,,	½	,,	179° F.
,,	,,	1	,,	212° F.
,,	,,	2	,,	249° F.
,,	,,	3	,,	273° F.
,,	,,	4	,,	291° F.
,,	,,	5	,,	306° F.
,,	,,	6	,,	319° F.

To find the Nominal Horse-power of Boilers. CORNISH BOILERS. Add the diameter of boiler and diameter of flue together in feet, multiply the sum by the length in feet, and divide the product by 8.

LANCASHIRE BOILERS. Add the diameters of both flues and the diameter of the boiler together in feet, multiply the sum by the length in feet, and divide the product by 8.

VERTICAL TUBULAR BOILERS. Add together the diameter of the shell, the diameter of the fire-box, and the diameters of all the tubes, all in feet; multiply the sum by the length in feet and divide by 12.

The actual horse-power of a boiler can be obtained by calculating the number of cubic feet of water evaporated per hour.

To find the Number of Cubic Feet of Water Evaporated per Hour. Multiply the number of square feet of water surface by the evaporation in inches of gauge glass, multiply the product by 5, and divide the result by the number of minutes occupied in evaporation.

The Lever Safety Valve.

Let P = pressure of steam in lb/sq in. less atmospheric pressure
L = length of lever from fulcrum to weight, in inches
W = weight of load in pounds
W' = weight of lever in pounds
W'' = weight of valve in pounds
D = distance of the centre of gravity of the lever from the fulcrum in inches
A = area of valve in square inches
I = length of lever between fulcrum and valve, in inches.

$$P = \frac{\dfrac{WL}{I} + \dfrac{W'D}{I} + W''}{A}$$

Weight of a Boiler. Weight in pounds = surface in square feet × 6 for plates ⅛ in. thick.

Fuel

Evaporative Power of Fuels. The following quantities of fuels will evaporate 9 lb water when raised to 212° F.

1 lb steam coal
2 lb dry peat
2½ lb dry wood
3¼ lb cotton stalks
3½ lb brushwood
3¾ lb straw
4 lb sugar-cane refuse

The British Thermal Unit is the quantity of heat required to raise 1 lb of water 1° F. when at its maximum density, i.e. from 39° to 40° F.

To find the Quantity of Water in Pounds Evaporated per Pound of Coal. Multiply the number of cubic feet of water evaporated per hour by 62·5, and divide the product by the quantity of coals consumed per hour.

Speeding of Machinery

Speeding by Pulleys

Let V = velocity or number of revolutions per minute of the driving pulley
v = velocity or number of revolutions per minute of the driven pulley
D = diameter in inches of driving pulley
d = diameter in inches of driven pulley.

Then—

$$v = \frac{VD}{d}; \quad V = \frac{vd}{D}; \quad d = \frac{VD}{v}; \quad D = \frac{vd}{V}$$

Speeding by Wheels

Let N = number of teeth in driving wheel
n = number of teeth in driven wheel
V = velocity or number of revolutions of driving wheel per min.
v = velocity or number of revolutions of driven wheel per min.

Then

$$v = \frac{NV}{n}; \quad V = \frac{nv}{N}; \quad n = \frac{NV}{v}; \quad N = \frac{nv}{V}$$

Safety Precautions for Boilers and Machinery

All boilers and machinery operated by steam, compressed air, oil, gas, or electricity must be in charge of a competent man.

All boilers, compressors, and machinery should be kept in a fit state and condition for work. Every boiler and pressure vessel must be provided with safety valves and gauges. Every boiler must be thoroughly cleaned at least once in six months, and every twelve months be subjected to an examination and hydrostatic test. The test of working boilers should be equal to one and a quarter times at which the safety valve blows off; and that of new boilers one and a half times the intended working pressure. Air receivers should be tested once in twelve months to one and a quarter over the allowed working pressure.

If two or more boilers are connected to the main steam pipe, and one is to be cleaned, the latter should be cut off from the main pipe, and the valve locked.

Steam gauges should be tested and adjusted once in twelve months, and no gauge should be passed which varies more than five pounds from the standard gauge. All dangerous parts of machinery must be kept securely fenced. All persons whose duty it is to attend moving machinery on the surface should wear close-fitting and close-fastened garments which cannot be readily entangled in such machinery. Any person who is in charge of machinery in use at a mine must be registered as the holder of a certificate of competency or service as an engine driver granted by a recognized authority.

The engine driver of a hoisting engine should see that the engine, drums, valve gear, brakes, indicators, signal bells, and other gear under his charge are in good order before taking over. The engine driver should be thoroughly acquainted with the signals in use. Should a signal be indistinct, he must not start the engine until it has been repeated and he thoroughly understands it.

If winding has been suspended for more than four hours, the engine driver should run the cage up and down at least once before commencing to raise and lower men.

Only authorized persons should be allowed inside a hoisting engine room, and they must not enter while the engine is in motion. Windows of the engine room should be sufficiently high off the ground to prevent the attention of the engine driver being distracted by outside events. The driver must have a clear view of the mouth and brace of the shaft.

Every hoisting machine should be provided with adequate brake, and an indicator independent of any mark on the rope.

Lubricants

The requirements of a lubricant are as follows. They must have sufficient body at the probable temperature to which they are likely to be subjected so as to prevent them from being squeezed out by pressure between bearing surfaces. They must not decompose or evaporate at ordinary working temperature. They must not contain acid which might injure the bearing surface. They must be sufficiently fluid. The coefficient of friction must be low. They must not freeze at any temperature to which the bearing may be exposed.

Ordinary lubricants are animal fats, vegetable and mineral oils, flake graphite, soapstone, and powdered mica. Animal fats and oils decompose at moderate temperatures and set acids free. Vegetable oils are used where bearing pressure is light and the temperature low. Mineral oils can be obtained to suit practically all conditions of lubrication. Greases are a mixture of mineral oil with animal or vegetable soap, useful for slow-moving bearings under heavy pressure. Solid lubricants are generally mixed with oil or grease, but are not suitable for ball-bearings.

Flash point is the temperature at which an oil will give off enough vapour to ignite and flash when a flame is held over the surface. Burning point is the temperature at which an oil will give off enough vapour to ignite and continue to burn.

For engine cylinders using high pressure steam, high-grade cylinder oil should be used. For engine cylinders using steam below 100 lb pressure second-grade cylinder oil can be used. Air compressors and high-speed internal combustion engines need a liquid base oil with a relatively low viscosity and a high flash point. For slow moving bearings under heavy pressure use grease. If the pressure is exceptionally heavy, add graphite to the grease.

CHARACTERISTICS OF LUBRICATING OILS

Kind of Oil	Specific Viscosity 70° F	Specific Gravity Water -1	Flash-point ° F	Burning point ° F
Heavy engine and machine .	185	0·875	400	445
Ordinary engine and machine .	180	0·873	410	460
Light machine . . .	100–150	0·870	400	440
High-grade cylinder . .	175–200	0·903	600	650
Ordinary cylinder . . .	185	0·899	570	615
Gas engine	300	0·893	320	350
Automobile cylinder . .	195	0·877	430	485

Embankments

APPROXIMATE SHRINKAGE OF EMBANKMENTS
(Trautwine)

Gravel or sand . . .	8 per cent
Clay	10 ,,
Loam	12 ,,
Loose vegetable surface soil .	15 ,,
Puddle clay . . .	25 ,,

ANGLES OF REPOSE AND FRICTION OF EMBANKMENT MATERIALS (Fanning)

Material	Angle of Repose	Coef. of Friction	Ratio of Slope Hor. to 1 Vert.
Dry fine sand	28°	0·532	1·88
Dry coarse sand	30°	0·577	1·73
Damp clay	45°	1·000	1·00
Wet clay	15°	0·268	3·73
Clayey gravel	45°	1·000	1·00
Shingle	42°	0·900	1·11
Gravel	38°	0·781	1·28
Firm loam	36	0·727	1·38
Vegetable soil	35°	0·700	1·43
Peat	20	0·364	2·75
Masonry on clayey gravel	30	0·577	1·73
Masonry on dry clay	27	0·510	1·96
Masonry on moist clay	18°	0·325	3·08
Earth on moist clay	45	1·000	1·00
Earth on wet clay	17°	0·306	3·26

Wind

Velocity in m.p.h.	Velocity in Ft/sec	Pressure in Lb/sq ft	Remarks
1	1·467	0·005	Hardly perceptible
2	2·933	0·020	Pleasant
3	4·400	0·045	,,
4	5·867	0·080	,,
5	7·33	0·125	,,
10	14·67	0·5	,,
12½	18·33	0·781	Fresh breeze
15	22·0	1·125	,,
20	29·33	2·0	,,
25	36·67	3·125	Brisk wind
30	44·0	4·5	Strong wind
40	58·67	8·0	High wind
50	73·33	12·5	Storm
60	88·0	18·0	Violent storm
80	117·3	32·0	Hurricane
100	146·7	50·0	Violent hurricane, uprooting large trees

Windmills make use of about 29 per cent of the energy imparted to them.

Mechanical Drawing

HINTS. When choosing *instruments* look well to workmanship. With bad instruments no good work can be done.

Bowpens should be broad, thick, and strong, and not much bowed—the points should be rounded off.

Needle-pointed instruments are much to be preferred.

Never draw more pencil lines than necessary.

Accuracy is all important, small errors are multiplied in long construction.

Always ink-in circles and curves before straight lines.

When many concentric circles are required, use a horn centre, to avoid making a big hole in the paper and consequent false lines.

Never leave a pen with ink between the nibs to dry.

Use bread in preference to india-rubber, especially when cleaning the whole surface of a drawing.

Shade lines are drawn *outside* the outline of the object, and *dimensions* are measured from *inside* the shade line.

When shading or shade lines are employed, the light is supposed to come over the left shoulder.

Shade lines should never be drawn to indicate the outline or contour of a curved surface.

COLOURS. *Illuminated* plane surfaces *parallel* to the plane of projection receive flat tints. The nearer the surface is to the eye the lighter such tints should be.

Illuminated plane surfaces *inclined* to the plane of projection receive graduated tints, becoming *darker* as such surfaces recede from the eye.

Unilluminated plane surfaces *parallel* to the plane of projection receive flat tints. The nearer the surface is to the eye the darker such tints should be.

Unilluminated plane surfaces *inclined* to the plane of projection receive graduated tints becoming *lighter* as the surface recedes from the eye.

Shading in colour is done in strips, softening the edges each time, *beginning* always with the *darkest* portions of the object, and washing over and beyond the previously coloured portion each time.

The following colours are in general use to represent the more common materials—

1. *Cast Iron.* Payne's Grey or mixture of indigo, sepia, and lake.
2. *Wrought Iron.* Prussian blue.
3. *Steel.* Prussian blue and crimson lake.

4. *Brass.* Elevation . Gamboge.
Section . Dark Indian yellow.
5. *Copper.* Gamboge and crimson lake.
6. *Ordinary building.* Sepia and yellow ochre.
7. *Brick (common).* Elevation . Light red.
Section . Crimson lake.
8. *Brick (fire).* Elevation . Light red and yellow ochre.
Section . Ditto with crimson lake.
9. *Wood.* Elevation . . Yellow ochre.
Section . . Burnt sienna.
10. *Earth.* Burnt umber.
Lines. Red . . Carmine.
Blue . French ultramine.

Cast shadows are seldom used, but if desired they should be washed in neutral tint or Indian ink before colouring.

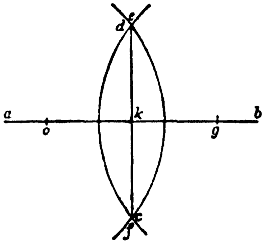


FIG. 81

In *Mine plans* the spaces left by the removed ore masses are coloured purple for tin; green for copper; blue for lead. The adit level is coloured blue and the levels below it red, green, yellow, violet, and brown in succession.

SURVEY SCALES. The scales usually employed for the plans of metalliferous mines are 4 or 8 fathoms to the inch, sometimes 5 or 10 fathoms. For colliery plans scales of 2 or 3 chains to the inch, or of 25·34 inches to the mile, are the most usual. In the constriction of maps of the Ordnance Survey of Great Britain, the following scales are used—

Towns	1 : 500	or 126·72 inches to the mile
Parishes	1 : 2500	25·34 ,, ,,
Counties	1 : 10560	6 ,, ,,
The Kingdom	1 : 63360	1 ,, ,,

To divide a given line ab into two equal parts. From its ends *a* and *b* as centres, and with any radius greater than half *ab*, describe the arcs *c* and *d*, cutting each other at *e* and *f*; join *e* and *f*; *ab* is bisected at *k*.

If *ab* is very long first lay off equal distances *ao*, *bg* from *a* and *b*, and then proceed as if *og* were the line to be divided.

To divide a given line into any number of equal parts. Let *mn* be the given line, from *m* draw *mo* of indefinite length, and from *m* along *mo* step off the required number of equal parts, say 6, join *n*6 and draw lines parallel to *n*6 through 5, 4, 3, 2, 1, and cutting *mn*, when *mn* will be divided into six equal parts.

To divide a given line into two parts which shall have a given proportion to one another. This is done on the same principle as the last: thus let the proportion be as 1 to 2. First draw any line *mo* (Fig. 82) and with any convenient opening of the dividers make *mx* equal to one step; and *xz* equal to two steps. Join *zn* and draw *xy* parallel to *zn*. Then *my* is to *yn* as 1 is to 2.

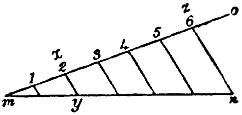


FIG. 82

To draw an ellipse. An ellipse (Fig. 83) may be quickly and fairly accurately drawn thus: Given the major and minor axes AB, CD. With C and D as centres and radius equal to AF (semi-major axis) drawing arcs cutting AB in E and G. Stick in two ordinary pins, through the paper and into the drawing board, at E and G and tie to them a piece of silk or thread of

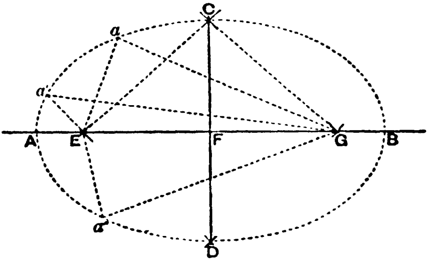


FIG. 83

such a length that it equals EC + CG. If the point of a pencil be carried round in this loop so the thread will take up such positions as E*a*G, E*a'*G, E*a''*G, etc., the pencil point will trace an ellipse having the major and minor axes AB and CD.

Copying Drawings. To trace on to ordinary drawing paper, stretch the paper over the drawing and saturate it with benzine by means of a cotton pad; this makes it transparent so that the drawing below may be traced. Pencil, Indian ink, and water colours, take equally well on the benzined surface. When the copy is completed the benzine rapidly evaporates and leaves the paper in its original condition. Large drawings only have benzine applied to a portion at a time.

Blue prints or cyanotypes: giving white lines on blue ground—

Sir John Herschel's	Marion's
140 gr ferric ammonic citrate	9⅜ oz ferric ammonic citrate
120 gr potassic ferri-cyanide	6¼ oz. potassic ferric oxide
2 oz distilled water	Dissolve separately in pure water, and make up to 1 quart

Willis's platinotype process, giving white lines on black ground—

Sensitizing solution	60 gr potassic-platinous chloride 60 gr ferric oxalate 1 oz water

Expose until the paper acquires a dull orange tint.

Developing solution	130 gr potassic oxalate 1 oz water

Develop for 4 seconds at temp. of 150 to 200° F., then wash for 10 minutes in 1 pt hydrochloric acid with 60 pt water, and finally for 15 minutes in relays of fresh water.

Morse Signals

These signals can be made by tapping, with one or two flags, or with lights. The alphabet never has more than four elements: the numerals always have five.

Dot Letters		*Dash Letters*	
E	·	T	—
I	· ·	M	— —
S	· · ·	O	— — —
H	· · · ·		

Short Opposites				*Long Opposites*			
A	· —	— ·	N	G	— — ·	· — —	W
U	· · —	— · ·	D	F	· · — ·	· — · ·	L
V	· · · —	— · · ·	B	P	· — — ·	— · · —	X
R	· — ·	— · —	K	Q	— — · —	— · — —	Y

Letters with no Opposites

C	— · — ·
J	· — — —
Z	— — · ·

Morse Numerals

1	· — — — —	6	— · · · ·
2	· · — — —	7	— — · · ·
3	· · · — —	8	— — — · ·
4	· · · · —	9	— — — — ·
5	· · · · ·	0	— — — — —

Miscellaneous Tables

A bar on a signal in Morse means that the signal is sent as one sign, not as two or more letters.

Semaphore	*Morse*	*Meaning*
A	T	General answer
AAA	AAA	Full stop
AR	AR	End of message
ii	ii	Break sign
iii	iii	Decimal point
J	No sign in Morse so long as numerals are sent	Alphabetical sign
VE	VE	Call up sign
C	C	You are correct
GB	GB	Good-bye
MH	MH	Move higher (or further away)
ML	ML	Move to your left
MO	MO	Move lower (or closer in)
MR	MR	Move to your right
Q	Q	Wait a minute
R	R	Message received
RU	RU	Who are you?
WA	WA	Word after
WB	WB	Word before
G	G	Repeat back
IMI	IMI	Repeat sign
K	K	Go on
MM	MM	Separating whole numbers from fractions in the same group

Semaphore	*Morse*	*Meaning*
NR	$\overline{\text{NR}}$	Horizontal bar to separate the numerator from the denominator of a fraction
V	V	From
VA	$\overline{\text{VA}}$	Communication finished
Numeral sign	FI	Figures coming
Alphabetical sign	FF	Figures finished, letters coming
Opposite to L	8 dots	Delete a word sent incorrectly

Whatever the duration of a dot, the dash takes three times as long. A letter must be made continuously from start to finish without any interval between the elements composing it, otherwise it might be mistaken for two or three other letters. An interval equal to one dash is observed after each letter.

When flags are used, the signaller must stand facing or with his back to the distant station according to the direction of the wind. The flag is waved in a vertical plane, not swept round to the front. All motions of the flag must be sharp. The flag must be kept fully exposed when sending: at other times it must be completely hidden from the view of the distant station.

Safety Precautions. Signals in the same country should be the same in all mines, otherwise men going from one mine to another where a different code of signals is in use may cause an accident.

Shaft signals should be printed and posted up at the top of the shaft and at each plat.

Every underground road on which persons travel and in which the haulage is by means of ropes should be provided, if exceeding 30 yd in length, with some proper means of communicating distinct and definite signals between the stopping places of such haulage and the ends of the road.

The voltage used for the electric signals must not exceed 25 volts. When bare conductors have to be handled for the purpose of signalling a system of automatic excess voltage protection, where practicable, should be installed, and so arranged that the supply likely to cause any excessive voltage will be disconnected when the voltage between the signalling system and earth exceeds 25 volts. The conductors should be kept as far away as possible from lighting or power cables.

Income Table

INCOME TABLE FOR THE YEAR, QUARTER, CALENDAR MONTH, WEEK, OR DAY

Yearly			Quarterly			Monthly			Weekly			Daily		
£	*s.*	*d.*	£	*s.*	*d.*	£	*s.*	*d.*	£	*s.*	*d.*	£	*s.*	*d.*
6	0	0	1	10	0		10	0		2	3¾			4
6	10	0	1	12	6		10	10		2	6			4¼
7	0	0	1	15	0		11	8		2	8¼			4½
7	10	0	1	17	6		12	6		2	10½			5
8	0	0	2	0	0		13	4		3	0¾			5¼
8	10	0	2	2	6		14	2		3	3			5½
9	0	0	2	5	0		15	0		3	5½			6
9	10	0	2	7	6		15	10		3	7¾			6¼
10	0	0	2	10	0		16	8		3	10			6½
10	10	0	2	12	6		17	6		4	0¼			7
11	0	0	2	15	0		18	4		4	2¾			7¼
11	10	0	2	17	6		19	2		4	5			7½
12	0	0	3	0	0	1	0	0		4	7¼			8
12	10	0	3	2	6	1	0	10		4	10½			8¼
13	0	0	3	5	0	1	1	8		5	0			8½
13	10	0	3	7	6	1	2	6		5	2¼			9
14	0	0	3	10	0	1	3	4		5	4¼			9¼
14	10	0	3	12	6	1	4	2		5	6¾			9½
15	0	0	3	15	0	1	5	0		5	9			10
15	10	0	3	17	6	1	5	10		5	11¼			10¼
16	0	0	4	0	0	1	6	8		6	1¾			10½
16	10	0	4	2	6	1	7	6		6	4			11
17	0	0	4	5	0	1	8	4		6	6¼			11¼
17	10	0	4	7	6	1	9	2		6	8½			11½
18	0	0	4	10	0	1	10	0		6	11			11¾
18	10	0	4	12	6	1	10	10		7	1¼		1	0
19	0	0	4	15	0	1	11	8		7	3½		1	0½
19	10	0	4	17	6	1	12	6		7	5¾		1	1
20	0	0	5	0	0	1	13	4		7	8		1	1¼
30	0	0	7	10	0	2	10	0		11	6		1	7¾
40	0	0	10	0	0	3	6	8		15	4½		2	2¼
50	0	0	12	10	0	4	3	4		19	2¾		2	9
60	0	0	15	0	0	5	0	0	1	3	1		3	3½
70	0	0	17	10	0	5	16	8	1	6	11		3	10
80	0	0	20	0	0	6	13	4	1	10	9¼		4	4¾
90	0	0	22	10	0	7	10	0	1	14	7¼		4	11½
100	0	0	25	0	0	8	6	8	1	18	5½		5	6
200	0	0	50	0	0	16	13	4	3	16	11		10	11¾
300	0	0	75	0	0	25	0	0	5	15	4		16	5¾
400	0	0	100	0	0	33	6	8	7	13	10	1	1	11¾
500	0	0	125	0	0	41	13	4	9	12	3½	1	7	5¾

Moneys of Foreign Countries

Mining engineers are often engaged to carry out work in foreign countries. If paid in foreign currency, they should inquire whether it is gold or silver, whether the money can be taken out of the country at the termination of their engagement, and at what rate of exchange.

Country	Monetary Unit	Country	Monetary Unit
Argentina	Peso	India	Rupee
Austria	Schilling	Italy	Lira
Belgium	Franc	Japan	Yen
Bolivia	Boliviano	Liberia	Dollar
Brazil	Cruzeiro	Mexico	Peso
British Honduras	Lempira	Newfoundland	Dollar
Bulgaria	Lev	Nicaragua	Cordoba
Canada	Dollar	Norway	Krone
Chile	Peso	Panama	Balboa
China	Yuan	Paraguay	Peso
Colombia	Peso	Persia	Rial
Costa Rica	Colon	Peru	Sol
Cuba	Dollar	Philippine Is.	Peso
Czechoslovakia	Koruna (Crown)	Poland	Zloty
		Portugal	Escudo
Denmark	Krone	Rumania	Leu
Ecuador	Sucre	Russia	Rouble
Egypt	£E (100 piastres)	Salvador	Colon
		Spain	Peseta
Finland	Markka	Straits Settlements	Straits Dollar
France	Franc		
Germany	Mark	Sweden	Krona
Great Britain	Pound	Switzerland	Franc
Greece	Drachma	Turkey	£T
Guatemala	Quetzel	United States of America	Dollar
Haiti	Gourde		
Holland	Gulder	Uruguay	Peso
Honduras	Lempira	Venezuela	Bolivar
Hungary	Pengo	Yugoslavia	Dinar

PERPETUAL CALENDAR FOR ASCERTAINING THE DAY OF THE WEEK

Table of Dominical Letters

Year of the Century				Centuries: 1700, 2100	1800, 2200	1900, 2300	2000, 2400
0	28	56	84	C	E	G	A
1	29	57	85	B	D	F	G
2	30	58	86	A	C	E	F
3	31	59	87	G	B	D	E
4	32	60	88	E	G	B	C
5	33	61	89	D	F	A	B
6	34	62	90	C	E	G	A
7	35	63	91	B	D	F	G
8	36	64	92	G	B	D	E
9	37	65	93	F	A	C	D
10	38	66	94	E	G	B	C
11	39	67	95	D	F	A	B
12	40	68	96	B	D	F	G
13	41	69	97	A	C	E	F
14	42	70	98	G	B	D	E
15	43	71	99	F	A	C	D
16	44	72		D	F	A	B
17	45	73		C	E	G	A
18	46	74		B	D	F	G
19	47	75		A	C	E	F
20	48	76		F	A	C	D
21	49	77		E	G	B	C
22	50	75		D	F	A	B
23	51	79		C	E	G	A
24	52	80		A	C	E	F
25	53	81		G	B	D	E
26	54	82		F	A	C	D
27	55	83		E	G	B	C

Month	Dominical Letter						
Jan. Oct.	A	B	C	D	E	F	G
Feb. Mar. Nov.	D	E	F	G	A	B	C
Jan. Apr. July	G	A	B	C	D	E	F
May	B	C	D	E	F	G	A
June	E	F	G	A	B	C	D
Feb. Aug.	C	D	E	F	G	A	B
Sept. Dec.	F	G	A	B	C	D	E

1	8	15	22	29	*S*	S	F	Th	W	Tu	M
2	9	16	23	30	M	*S*	S	F	Th	W	Tu
3	10	17	24	31	Tu	M	*S*	S	F	Th	W
4	11	18	25		W	Tu	M	*S*	S	F	Th
5	12	19	26		Th	W	Tu	M	*S*	S	F
6	13	20	27		F	Th	W	Tu	M	*S*	S
7	14	21	28		S	F	Th	W	Tu	M	*S*

EXPLANATION. Under the *Century*, and in the line with the *Year of the Century*, is the Dominical Letter of the Year. Then in the line with the month find the column containing this letter; in this column, and in the line with the day of the month, is the day of the week. In Leap-years, the letters for January and February are in the lines where these months are printed in *italics*.

EXAMPLES. For Dec. 31st, 1871: for 1871 the letter is A; under A, in a line with 31, is Sunday; and for *Jan.* 1st, 1872, the letter is F; under F, and in a line with 1, is Monday.

Proof Readers' Signs

ʎ	Insert
δ/	Delete
୭	Turn the other way up
trs/	Transpose
w.f.	Wrong fount
#	Insert more space
⁀‿	Close up
┐	Move to the right
┌	Move to the left
×	Change bad letter
⊥	Push down space
Out see copy	Something left out. See copy
//	Straighten lines
Cap	Use capital letter
Caps or ≡	Set in capitals
s.c. or ═	Use a small capital or small capitals
⊓	Move up
⊔	Move down
〰	Bold face
l.c.	Use lower case
ital	Use italics
rom	Use roman
○/	Insert a full stop
,/	Insert a comma
⊙/	Insert a colon
;/	Insert a semi-colon
’/	Insert an apostrophe
”/	Insert quotation marks
-/	Insert a hyphen
/—/	Insert a dash
□	Insert an em-quad
n.p.	Begin a new paragraph
run on	Do not begin a new paragraph
stet or	Disregard correction
Qy.	Query. Is this right?
⌒	Print as a diphthong

GLOSSARY

Acid. Acids are compounds which undergo electrolytic dissociation in aqueous solutions with the formation of hydrogen ions.

Adiabatic. When no provision is made for cooling air during compression.

Adit. An underground drift, the entrance to which is at the surface, which serves as a drain.

After-damp. The gases resulting from an explosion of fire-damp, mostly composed of carbon dioxide, carbon monoxide, and nitrogen.

Air compartment. An air-tight portion of a shaft or winze used for ventilation purposes.

Air course. Any underground passage for the special purpose of ventilation.

Air crossing. An air course which conveys air over another air course.

Air door. A door used for regulating currents of air in underground workings.

Air-lock. A space between two doors into which one can pass when going from one air current to another without upsetting the ventilation.

Air sollar. An air-tight wooden decking in a drift, a sufficient height above the rock bottom to allow water to drain out and cool air to pass in.

Air stack. A chimney connected with an upcast shaft to assist the draught.

Alkali. A soluble hydroxide which turns red litmus blue or methyl orange pink to yellow.

Alloy. A homogeneous mixture of two or more metals by fusion.

Alluvial. Minerals in the form of sand, gravel or shingle.

Amalgam. An alloy of quicksilver with some other metal.

Amortization. The redemption of capital invested in a plant with accumulated interest thereon.

Amygdaloidal. Almond-shaped.

Analysis. All methods for the determination of and/or estimation of the constituents of a compound or mixture.

Anemometer. An instrument for measuring the velocity of an air current.

Angle beam. A two-limbed beam used for turning the direction of pump rods in a shaft.

Anhydrous. When a compound is deprived of all its water.

Anneal. To toughen certain metals by heating and then allowing them to cool slowly.

Anticlinal. When rocks are bent over in the form of an arch.

Antimony, crude. The mineral antimonite sweated out from its gangue.

Antimony, star. The metal antimony when crystallized and showing fern-like markings on the surface.

Aqua fortis. Nitric acid.

Aqua regia. A mixture of hydrochloric and nitric acids.

Archean. Rocks of a transition period between the original crystalline rocks and ordinary sedimentary rocks, such as crystalline schists.

Arenaceous. Sandy.

Argillaceous. Clayey.

Argol. Crude tartar deposited from wine.

Arrastra. A large circular trough in which drags are pulled round. Used for grinding and amalgamating gold ores.

Artesian well. A bore hole put down to tap subterranean water subject to hydraulic pressure.

Assay. A metallurgical term for the quantative detection of metals present in ores and metallurgical products.

Assay ton weight. A system made up from a comparison of the avoirdupois, troy, and gramme weights, which is useful by saving much calculation.

Azimuth. An arc of the horizon between the meridian of a place and a vertical circle passing through the centre of a celestial body.

Azoic. The age of rocks that were formed before animal life existed on this earth.

Back shift. The afternoon shift.

Back stay. A wrought-iron forked bar attached to the rear of a mine car when ascending an incline, so as to throw it off the rails in case the hauling rope or coupling gives way.

Backs. That portion of a lode overhead which has not been extracted.

Bal. Cornish for a mine.

Balance box. A large box placed on one end of a balance bob filled with old iron or stones to counterweight the pump rods of a Cornish pump.

Balance brow. A self-acting incline in a steep coal seam on which a platform on wheels travels and carries a tub of coal.

Band. A thin layer of rock in a seam of coal.

Bank. The surface about the mouth of a pit.

Banket. Auriferous conglomerate of South Africa.

Bar. A ridge of rock or sand formed in the bed of a river.

Barney. A small car used on an inclined plane attached to a rope for pushing a mine car up the slope.

Barrier pillar. A solid wall of coal left between adjoining collieries, between panels of the same colliery, or between workings under a body of water and under dry land; it serves as security in case of fire or flooding.

Base metal. A metal not classed with the precious metals (gold, silver, platinum, etc., which are not easily oxidized)

Basque or Brasque. A mixture of clay and coke or charcoal, used for the bottom of furnaces.

Batea. A wooden bowl used for panning off gold or tin ore.

Battery. A set, such as a set of stamps for crushing purposes.

Beach-combing. Working the sand of a beach for gold, tin, or platinum.

Bear. The accumulation of iron at the bottom of a blast-furnace.

Bearers. Beams of timber let into the sides of a shaft to act as supports for sets of timber.

Bearing. (1) The course of a compass. (2) Supports for a beam, shafting, etc.

Bearing-up pulley. A pulley fixed in a frame so arranged as to tighten or take up the slack of an endless belt.

Bed. A layer of one kind of rock.

Bed plate. A large plate of iron laid on a foundation for something to rest on.

Bed rock. The rock on which an alluvial deposit has been laid.

Bell-sheave. A sheave in the shape of a truncated cone used in connection with the main- and tail-rope system of haulage at curves so as to keep the rope close to the ground.

Bench mark. A mark made by surveyors for future reference.

Bessemer steel. Formed by forcing air into a mass of molten cast-iron, by which means the excess of carbon present is separated from it until only enough remains to constitute cast-steel.

Bind or Batt. Indurated shale, often forming the roof of a coal seam.

Black band. Carbonaceous ironstone mingled with coaly matter sufficient for its own calcination.

Black damp. Mostly nitrogen associated with carbon dioxide.

Black jack. Strictly speaking a dark variety of zincblende; but may be applied to any near-black material.

Black lead. Graphite.

Black sand. Heavy dark minerals which accompany gold in alluvial deposits, such as cassiterite, chromite, ilmenite, magnetite, and tourmaline.

Black tin. Dressed cassiterite or tin stone.

Blanket-table. An inclined table covered with blankets to retain the heavier particles of minerals fed on to it.

Blind coal. Natural coke.

Blind lode. A lode that has no visible outcrop.

Blind shaft. A shaft that does not come to the surface.

Block reefs. Veins showing frequent contractions and expansions.

Blocking-out. A method of working deep leads.

Blow. A large increase in the size of a lode.

Blower. A sudden emission of gas from a coal seam.

Blow-in. To commence a smelting process.

Blown-out hole. When a charge is unable to do the required work and only the tamping is blown out.

Blue john. Blue coloured fluorspar.

Blue metal. A name applied locally to various rocks, such as basalt, used for road-making.

Blue stone. Sulphate of copper.

Booming. A method of ground sluicing by emptying the contents of a reservoir suddenly on material it is desired to wash.

Boot. The bottom of an elevator for raising ore.

Bord. A heading 3 to 5 yd. wide, separated from another by a pillar of coal which is mined later.

Bort. An imperfect diamond of no use as a gem.

Bosh. The plane in a blast-furnace where the greatest diameter is reached.

Botryoidal. Grape-like in appearance.

Bottom gas. A mixture of explosive gases sometimes collected in depressions in coal mines.

Brace. A permanent platform on the headframe of a shaft on which men work.

Brace head. A handle for raising and turning the rods of a boring plant.

Brass. Iron pyrites associated with coal.

Brattice. A partition in a drift or shaft for ventilation purposes.

Break-through. A narrow passage cut through a coal pillar to connect adjoining bords.

Breast. The stall or bord in which coal is mined.

Breccia. A rock consisting of angular pieces cemented together.

Breeze. Fine coke.

Bridge. The horizontal pillar of ore left between the top of a stope and the floor of the drift above.

Bridge rails. Rails made in the form of an inverted U, generally in short lengths, which are light to handle and can be brought within easy shovelling distance of the face.

Bridle chains. Short chains by which a cage is connected to the winding rope.

Briquette. A fuel made of fine coal pressed into the form of a brick.

Brown coal. A younger coal than bituminous, formed from a higher class of vegetation.

Brown stone. Oxidized iron pyrites.

Brush. To brush the roof is to take down some of the rock overhead to give more head-room.

Buck quartz. Massive non-metal bearing quartz.

Buck staff. Uprights for bracing a reverberatory furnace.

Bucket. The top valve or clack of a pump lifting set.

Bucket sword. A wrought-iron rod to which the pump bucket is attached.

Bucking hammer. A slightly curved piece of iron provided with a horizontal handle used for grinding up mineral samples by hand.

Buddle. An inclined round or oblong table on which finely-ground minerals are concentrated.

Buffer rope. Wire ropes placed between the track of cages in a shaft where rope guides are employed, so as to prevent the cages from colliding.

Bull. An iron rod used for ramming clay into the sides of a watery drill hole.

Bull wheel. A wheel used in rope boring on which the rope is coiled.

Buller shot. A second hole drilled close to a miss-fire and exploded to do the work the first shot missed.

Buntons. Timbers placed horizontally across a shaft to carry cage or skid guides, pump rods, rising main, etc.

Burning point. The temperature at which an oil will give off enough vapour to ignite and continue to burn.

Burrow. A waste heap

Butt. Joints in coal at right-angles to the main joints or face.

Cage. A platform on which men or mine cars can travel up and down in a shaft.

Cage guides. Long pieces of wood, rails, or steel rope fixed in a shaft to guide cages.

Cainozoic. The Tertiary period.

Caking coal. Coal that agglomerates when heated to form coke.

Calcareous. Containing lime.

Calcine. To heat a substance sufficiently to drive off its volatile contents, but not sufficiently to melt it.

Californian pump. A rough pump made out of a wooden tube through which an endless belt with flights attached circulates. Used for pumping water from shallow ground.

Campaign. The length of time a furnace remains in blast.

Canch. The thickness of stone to be removed from the roof or floor of a gangway to improve the gradient of the road.

Candle power. The amount of light given off by a sperm candle which burns at the rate of 120 grains per hour.

Cannel coal. This has a smooth conchoidal fracture; has little or no structure; is rich in volatile hydrocarbons, and will burn like a candle.

Cap. (1) The pale bluish elongation of the flame of a lamp in the presence of fire-damp. (2) The upper portion of a lode. (3) A stick of timber placed on a prop or legs.

Car. A truck used in mines.

Carat. A weight used in weighing gems equal to about four grains.

Carbonado. Black diamonds, mostly used for diamond drills.

Carbonaceous. Containing carbon.

Cartridge. A cylindrical shell for containing blasting material, made to fit the hole to be fired.

Case hardened. To convert the outer surface of wrought-iron into steel by heating it while in contact with charcoal.

Casing. Tubing inserted in a hole to keep the sides from falling in, or to hold back water.

Cast iron. Iron resulting from the reduction of iron ores which retains carbon up to 5 per cent, also silicon, sulphur, phosphorus, etc.

Cavils. Lots drawn by coal miners periodically for working places.

Cement. Consolidated auriferous gravel.

Cementation. The process of converting wrought-iron into steel by heating it in contact with charcoal; this produces blisters on the steel bars, hence blister steel. When the blisters are removed and the steel made more compact by reheating and subjecting it to a tilt hammer, it is called tilted or shear steel. If the blister steel is broken up, resmelted and run into ingots, it is known as ingot steel, which is harder and closer grained than tilted steel.

Chain. A measure 66 ft long divided into 100 links.

Chain-breast machine. A coal-cutting machine so constructed that a series of cutting points attached to a circulating chain work their way for a certain distance under a seam. When the limit is reached, the machine is withdrawn and shifted to one side when another cut is put in.

Chain pillars. Pillars left to protect gangways and air courses with which they run parallel. They are broken up into a chain of smaller pillars by stentons and bords.

Chair. See Keeps.

Chambering. Enlarging the bottom of a deep drill hole so that it can hold a larger charge, by exploding a small quantity of powder in it.

Charge. (1) The material fed into a furnace at one time. (2) The amount of explosive used in one hole.

Cheese weights. The circular cheese-shaped weights used to keep guide ropes taut.

Chert. A silicious rock.

Chili bars. Bars of impure copper weighing about 200 lb, imported from Chili, corresponding to blister copper, and containing about 98 per cent copper.

Chilian mill. A roller mill for crushing ore.

Chill hardening. Giving a greater hardness to the outside of cast iron by pouring it into iron moulds which causes the skin of the casting to cool rapidly.

Chimney. A cylindrical shoot of ore at the junction of two veins.

Chinese pump. Similar to the Californian pump, but made entirely of wood.

Chloride. Chlorine chemically combined with a base.

Choke-damp. Carbon dioxide left after an explosion of fire-damp.

Chromate. Chromic acid combined with a base.

Churn drill. A long iron bar with a cutting end of steel, worked by raising it and allowing it to fall.

Chute. A passage, or hole in the ground for ore to pass through from an upper to a lower level.

Clack. A valve that is opened and shut by the force of water.

Claim. A portion of ground pegged out and held by virtue of a miner's right.

Clay band. Argillaceous iron stone.

Clay course. Gouge or dig often found on the sides of veins.

Claying bar. An iron bar used to force clay into the wall of a wet drill hole.

Clean-up. To collect all the quicksilver and amalgam about a battery after a crushing.

Cleat. The smooth vertical partings which occur in seams, more or less at right-angles to one another, one set usually being more pronounced than the other.

Cleavage. The property of splitting more readily in some directions than others.

Clinometer. An instrument for measuring vertical angles.

Coal. A solid fuel which occurs in seams, being the fossilized remains of organic matter.

Coal measures. Seams of coal with their attendant rocks.

Coal-puncher or pick-machine. A coal cutter of the reciprocating type used for undercutting and nicking coal.

Coal seam. A bed of coal.

Coarse metal. Copper matte obtained from the first smelting.

Cob. To break up with a hammer for hand sorting.

Cod-piece. A wooden fish plate used for connecting the segments of a walling-curb.

Coke. The fixed carbon and ash of a coal sintered together.

Coking coal. A bituminous coal capable of leaving a caked mass when heated in an oven so as to drive off the volatile hydrocarbons.

Collar. The collar of a shaft is the top frame of a shaft at the surface.

Colliery. A coal mine together with its plant and adjuncts.

Colour. Minute trace or individual specs of gold.

Column or rising main. The pipe conveying water from a mine pump to a higher level or to the surface.

Conchoidal. Shell-like, such as the curved fracture of flint.

Conformable. Strata are conformable when they lie over one another having the same dip.

Contact lode. A lode lying between two dissimilar rocks.

Contour. The line which bounds the figure of an object.

Copper plate. A sheet of copper which when coated with quicksilver is used for amalgamating gold.

Cord of wood. A pile of wood 8 ft by 4 ft by 4 ft or 128 cu ft.

Cornish pump. A plunger pump in which the motion is transmitted through a walking beam to the rods.

Costeaning. Trenching for a lode.

Counter lode. A vein running obliquely across the direction of the regular veins of a district.

Country rock. The rock in which a mineral vein occurs.

Cradle or rocker. A box with a sieve mounted on rockers for washing auriferous gravel and saving gold.

Creep. The gradual rising of the floor or sagging of the roof in mine workings; sometimes erroneously described as crush.

Creeper chain. A strong circulating chain in which every few feet a horn is inserted, which catches the axle of a skip and draws it up an incline.

Crevicing. Picking out specs of gold caught in crevices of rock in the bed of watercourses.

Crib. (1) A cast-iron or wooden ring on which the brick lining or tubbing of a shaft is built. (2) A structure composed of horizontal timbers laid crosswise in pairs, also known as a pig sty. (3) A miner's lunch.

Cribbing. Close timbering with logs as when lining a shaft.

Cropping. Portion of a vein as seen exposed above ground.

Cross-course. A vein lying more or less at right-angles to the veins of a district.

Cross-cut. A passage driven more or less at right-angles to a lode.

Cross-gateway. A road kept through the goaf of a coal mine which branches off at an angle from the main gateway.

Cross-head or jockey. A framework that runs on guides placed a few feet above a sinking bucket in order to prevent it swinging too violently.

Crystal. A solid of definite geometrical form which a mineral has assumed.

Crush. The breaking up of coal and the rock overhead when pillars or timbers give way.

Culm. Inferior anthracite, and the slack of smokeless coal.

Cundy or conduct. The passage under a roadway into which an endless rope passes out of the way at the end of its track.

Cupel. A cup made of bone ash for absorbing molten litharge.

Curb or crib. A ring made up of segments of wood or iron used in connection with the lining of shafts. Walling or wedging curbes wedged tightly in position are used as a foundation on which to start brick lining or tubbing. A capping or holding-down curb is the top curb of a section of tubbing.

Curtain. Bagging suspended from the roof of a drift to divert a current of air.

Cut-through. A connection between bords used for ventilation and travelling purposes.

Cutting down. To cut down a shaft is to increase its size.

Damp. Mine gases or gaseous mixtures, such as fire-damp and choke-damp.

Darg. The restriction of the output of coal by miners.

Dead man. A log buried in the ground to which a mooring or other rope is attached.

Dead roast. When all volatile matter in an ore is completely driven off.

Dead work. Exploratory work which is not directly productive.

Deficient place. A working place where men cannot make fair average wages and for which they are given extra pay.

Deputy. An underground official who sees to the general safety of a district and that the timbering is properly and sufficiently done.

Derrick. A crane in which the rope forming the stay can be let out or hauled in, thus altering the inclination of the jib.

Detaching hook. A self-acting mechanical arrangement for setting the winding rope free from a cage in case of overwinding.

Devil's dice. Cubes of limonite, pseudomorph after iron pyrites.

Dial. An instrument used for measuring angles in underground surveying.

Diamond. A pointed wooden or iron arrangement placed between rails just before a curve where skips are liable to be derailed. This enables them to mount the rails again.

Die. The bottom iron block of a battery or grinding pan on which the shoe works.

Diggings. Shallow alluvial workings for gold, etc.

Diluvium. Alluvial deposits of the Pleistocene period.

Dip. The angle which the slope of a stratum makes with the horizontal.

Dipping needle. A magnetic needle suspended in a vertical plane for locating iron deposits.

District. A certain area of a coal mine into which it is found advisable to divide a colliery for convenience in working.

Divining or dowsing rod. An indicator used by people possessed with certain faculties, for finding underground water.

Dog. A short bar of iron with both ends sharpened and bent at right-angles to the main portion; used for connecting two pieces of timber together, or as a grip as at the top of a ladder.

Dog's hind leg. A name given to the junction of a winze and a rise when they meet at an awkward angle.

Dolly. A pestle and mortar for breaking up stone; the pestle generally being attached to a spring pole by a rope.

Down cast. The intake through which fresh air is drawn or forced through workings.

Draftage. A deduction made from the gross weight of ore when transported, to allow for loss.

Drag-bar or Dog-iron. A short heavy iron bar used as a drag behind a car when ascending a slope to prevent it running back in case the rope breaks.

Draw. To draw the pillars in a colliery is to rob or mine the pillars in the second working.

Drawing a charge. To take a charge from a furnace.

Drawlift. A pump that receives its water by suction and does not force it above its head.

Dredging depth. The depth below the surface of the water and the bed rock.

Dress. To dress a mine is to make it look better than it really is to a prospective investor.

Dressing. Preparing poor or mixed ores mechanically for metallurgical purposes.

Drift. (1) Sometimes wrongly termed a drive or a level. A horizontal passage underground in or alongside a lode, as distinguished from a cross-cut. (2) Loose alluvial deposit.

Drill bar. In rope boring the iron rod placed next the drill.

Driving-cap. The cap screwed to the casing of a bore hole which receives the blow from a pile driver when sinking it.

Drop shaft. A shaft, the lining of which is put together at the surface and lowered as work proceeds.

Dropper. A branch vein dropping from a lode.

Drum. A revolving cylinder round which a rope is wound.

Drummy. The sound given off when loose rock is struck by a hammer.

Duck machine. An arrangement of two boxes, one working inside the other like a gasometer. The inner box has a valve at the top through which air can enter. The outer box has a pipe passing through it. The inner box is worked up and down, forcing air through the pipe to the place required.

Duff. Fine coal between three-eighths and three-sixteenths inch.

Dumb drift. A short passage in a coal mine connecting the return airway with the upcast some distance above a furnace so that the mine gases shall not pass over the fire, where furnace ventilation is used.

Dummy. A counterbalance used on jigs which runs on a narrow gauge line between the jig rails.

Dump. A heap of mullock as discharged from a mine.

Drusy. A small cavity in a rock lined with crystals.

Dyke. A vertical sheet or wall of an intrusive rock.

Eluvial. An accumulation of mineral matter through atmospheric agencies.

Elvan. A Cornish name for most dyke rocks of that county irrespective of their mineral composition, but at the present day is restricted to quartz porphyries.

End-on. When the breast of coal is driven at right-angles to the main cleat.

Engine plane. An incline on which one set of skips is hauled up hill by an engine, while the other set of skips runs down hill by gravity.

Entry. A main haulage road, gangway, or airway in a colliery.

Escarpment. A nearly vertical natural face of rock or soil.

Face. The extreme end of a drift or other mining excavation.

Face board. Boards placed against the face of a drift to hold back loose material.

Face-on. When the face of a breast or entry is parallel to the main cleats of a seam of coal.

False bedding. Although the general stratification of a formation is parallel, the laminations of a particular bed may be oblique to the main stratification. This is called false bedding.

False bottom. (1) A stratum on which pay dirt lies, but which has other layers of pay dirt below it. (2) A movable bottom to some apparatus.

False set. A temporary set of timber used in a drift until work is sufficiently far advanced to put in a permanent set.

Fan. A revolving machine for forcing air into or sucking it out of a mine.

Fan drift. A short tunnel leading from the top of an air shaft to the fan.

Fat or gas coal. Coal containing much volatile oily matter.

Fault. A fracture dislocating a rock, lode, or seam.

Feather. (1) A slightly projecting narrow rib of steel that fits in a groove in shafting and a corresponding groove in anything that surrounds and slides on a shafting such as a pulley or cam. (2) To feather out means to diminish gradually, like a wedge.

Feed. (1) The forward motion of rock drills or coal cutters either by hand or made automatically. (2) To supply a machine with material to be treated.

Fire. To set alight to the fuse for exploding a charge.

Fire-boss. An underground official who examines a mine for gas, and inspects safety lamps taken into a mine.

Fire-clay. Any clay that will withstand a great heat without vitrifying.

Fire-damp. A mixture mainly of methane and air in explosive proportions.

Fire-setting. The process of exposing very hard rock to intense heat so as to make it more easy to break down.

Fired-to. When there is a free face to which a shot can be fired, in distinction to shooting in the solid.

First working, or working in the whole. The opening up of a seam of coal.

Firsts. The best ore picked from a mine.

Fish. To join two beams or rails together by pieces fastened to their sides.

Fissure. An extensive fracture in a rock.

Fissure vein. A mineral deposit which occurs in a fissure.

Flash point. The temperature at which an oil will give off enough vapour to ignite and flash when a flame is held over the surface.

Flat sheet. Pieces of boiler plate fastened to a timber foundation; used on the brace, at plats, and the junction of drifts and cross-cuts, to facilitate the turning of cars where there are no rails.

Fleet angle. The angle made between the two ends of a winding drum as a base and the pit-head pulley.

Float gold. Gold in thin scales which float on water when panning off.

Floaters. See Shoad stones.

Flood gate. A gate to let out an excess of water from a water channel.

Flour gold. The finest alluvial gold.

Flouring. When mercury is broken up into fine globules, which being coated with some substance will not readily unite.

Flucan. A soft greasy clay found in the joints of veins.

Fluid mass. Water thickened with fine sand, to assist the floating of coal in jigs.

Flume. An artificial watercourse, generally of wood, used either to divert water from a stream so that its bed can be worked for gold, or to convey water to another place for sluicing purposes.

Flux. Minerals added to a charge for smelting to combine with the gangue of the ore so as to promote fusion.

Flying fox. A single carrying rope and a single hauling rope which can raise or lower a bucket. Used for raising or dumping mineral substances.

Fly-wheel. A heavy revolving wheel for equalizing the motion of machinery.

Foliated. Lamellar or leaf-like, such as molybdenite.

Foot piece. (1) A wedge of wood placed on the footwall against which a stull piece is jammed. (2) A piece of wood placed on the floor of a drive to support a prop or leg of timber.

Footwall. The lower boundary of a lode.

Force pump. A pump that forces water above its valves.

Forepoling. Driving poles over a set of timber so that their ends project beyond the last set in order to protect miners from roof falls. Also used when mining in loose material.

Fork. (1) A tool for separating coarse material from fine, as when alluvial mining, or fork filling coal. (2) A deep receptacle to enable a pump to extract the bottom water. A pump is said to be "in fork" when the water is so low that air is sucked through the wind-bore.

Fork filled. Coal filled into skips with a fork having the tines about 1¼ in. apart. This separates the bulk of the slack from the round coal which should not have more than 10 per cent of its weight of slack left in it.

Formation. A series of strata comprising those that belong to a single geological age.

Fossicker. A man who seeks ore in deserted places.

Fossil. The impression of organic remains which are left in mineral matter.

Frame set. The full set of timber that supports the walls of a drift or shaft.

Free milling. Ores that require no preliminary treatment by roasting or chemicals.

Furnace. A suitable heating apparatus for roasting or smelting ores.

Fuse. The connection between an explosive charge and the source of ignition.

Gad. A small steel wedge used for loosening jointy rocks.

Gangue. Valueless material in a lode.

Gannister. A hard compact fire-clay.

Gash vein. A wedge-shaped vein which occurs in limestone.

Gateway. A road through the goaf in longwall mining.

Geode. A nodular stone hollow in the centre.

Geordie turn-out. Rails for a turn-out from a heading to a bord made of bar-iron of square cross-section instead of ordinary T-rails; so that the same rails can be used for right or left by simply reversing the rails.

Geyser. A natural fountain of hot water and steam.

Glory hole. Ore worked on the surface in a conical hole and passed to a level below.

Gob or Goaf. That part of a colliery from which the coal has been extracted and the space left filled up more or less with waste.

Gob stink. The odour given off by incompletely burnt coal.

Gossan or Iron cap. The spongy ferruginous outcrop of a lode left after the soluble portion has been dissolved out.

Gouge or dig. The soft clay which sometimes occurs on the walls of a lode which can be gouged or picked out to make a place for the ore to be "fired to."

Grade. (1) The quality of a substance. (2) The amount of fall or inclination in a road or stream.

Grass. Ore is said to be "at grass" when it has been brought to the surface.

Gravel. Waterworn stones about the size of marbles.

Green coal. Coal freshly mined.

Greenstone. A general term employed to designate green coloured igneous rocks not easily determined on the field; such as diabase, diorite, andesite, and gabbro.

Greywacke. A coarse impure slaty quartzite frequently found in Palaeozoic formations.

Grizzly. A bar screen for separating large pieces of ore or coal from those of a smaller size.

Ground sill. A log laid on the floor of a drift for the legs of a set to rest on.

Ground sluicing. Washing alluvial loosened by pick and shovel in trenches cut out of the bed rock.

Grub stake. A mining outfit and supplies provided a prospector by a sleeping partner.

Grunching. Shooting in the solid.

Guano. A deposit composed chiefly of the excreta of sea birds in rainless tracts; or of bats in caves.

Gutter. (1) A small drainage channel. (2) The lowest part of a lead which contains most of the gold in an alluvial deposit.

Half-on. When the breast of coal is driven half way between the face and end cleats.

Hang fire. When a shot is delayed for some time before going off.

Hanging deals. Planks used to suspend a lower curb from the one above it in cases where backing deals are necessary.

Hanging wall. The upper boundary of a vein.

Headframe. The structure above a shaft built mainly to support the sheaves for winding ropes.

Head race. The ditch that brings water from its source to alluvial workings.

Heapstead. The entire surface plant about a shaft.

Heave. The lateral movement of a fault.

Hog. When the stem and stern of a bucket dredge tend to drop.

Hogback. A roll occurring in the floor of a seam but not in the roof, nearly cutting out the coal.

Holing. (1) A horizontal cutting in or below a seam of coal. (2) A connection made between two or more roads underground.

Horn silver. Cerargyrite or chloride of silver.

H-piece. That portion of a column pipe containing the valves of a Cornish pump.

Horse-power. A mechanical horse-power is the force required to raise 33,000 lb one foot in a minute.

Housing. A cover for certain machinery or other things that require protection from the weather.

Hungry looking. Applied to worthless looking stone.

Hydraulicing. Working auriferous gravel by hydraulic power.

Hydrocarbons. Chemical compounds of hydrogen and carbon.

Hydrometer. An instrument for ascertaining the specific gravity of a liquid.

Hygrodeik. A form of dry and wet thermometers with a graph to assist in calculations.

Igneous rocks. Rocks which have been in a more or less fused state.

Impregnations. An ore dispersed through a rock. Also known as a Stock.

Inbye. Towards the working face from the surface.

Inclined or Underlie shaft. A shaft sunk in or parallel to the underlie of a vein.

In fork. When a pump continues working after water has receded below the holes of the wind bore.

Ingot. A lump of cast metal.

Ion. An ion is a charged atom or group of atoms in solution which carry the electric current during electrolysis. Solutions always contain equivalent numbers of positive and negative ions.

Iron hat. See Gozzan.

Isothermal. When air is compressed at a constant temperature.

Jack hammer. A pneumatic drill held in the hand.

Jacket. An extra surface covering, such as a water jacket for a blast-furnace, or an outer lining for a shaft sunk in swelling ground.

Jars. Two long links used in rope boring.

Jenkin. A road cut in a pillar of coal running in the same direction as the bords on either side of it.

Jig. (1) A machine for concentrating fairly coarse ore by the pulsating movement of water imparted to it. (2) An underground gravity plane for conveying coal to a level below in a highly inclined seam.

Jockey. A self-acting apparatus designed to free a hoisting or hauling rope when the vehicle to which it is attached has reached a certain point.

Joint. Natural fractures in a rock confined to one bed. Mostly due to drying or cooling.

Jud. That portion of a working face loosened by undercutting and nicking.

Jump. (1) An up-throw or down-throw fault. (2) To take underhand possession of another's claim.

Jumper. A drill used for drilling rock by simply lifting and dropping it. One or both ends are sharpened.

Kata thermometer. An instrument for determining the cooling effect of the air.

Keeps or Chairs. A support provided in shafts for cages to rest on at stopping places.

Kerf. The undercut made while mining coal.

Key-stone. The centre stone of an arch.

Kibble. The bucket used for raising ore or broken rock in a shaft.

Kick-up. An appliance for emptying cars.

Kieve. A large wooden tub used for the final concentration of tin ore.

Kiln. A chamber sunk in the ground or built of stone or brick for heating minerals in.

Kindly looking country. A change in the barren portion of a lode that indicates ore will soon be struck.

Kip. A raised length of track on which skips are disconnected from a hauling rope, so arranged that the skips can gravitate as required to the bottom of the hoisting shaft for caging.

Knocker. A hammer so arranged that it strikes an iron plate at the mouth of a shaft when pulled by a cord, so that those below can signal to those at the surface.

Konimeter. An instrument for testing fine dust.

Laccolite. An igneous rock that has pushed its way up through fairly horizontal sedimentary rocks and spread out laterally, forming a large boss without reaching the surface.

Ladder-way. A particular passage in a shaft, winze, or pass, used for men to get to or from their work.

Lagging. Poles or slabs laid over the outside of regular shaft or drift frame sets to hold back loose material.

Lander or Braceman. The man who receives a load of ore at the top of a shaft.

Laths. See Slabs.

Leaching. To dissolve something out with a liquid.

Lead (pronounced leed). A bed of alluvial pay dirt.

Leader. A small vein, supposed to lead to a larger one.

Leading winning. A heading going in advance of the ordinary bords.

Leg-piece. An upright stick of timber erected against the side of a drift to support one end of a cap piece.

Level. The horizon at which an ore body is opened up; often confused with a drift which is driven at a level.

Lift. A slice taken off a pillar when winning it.

Lifting guard, or Bird cage. Fencing placed round the mouth of a shaft which is lifted out of the way by a cage as it reaches the surface.

Lignite. Brown coal showing a woody structure.

Linked veins. A series of branch veins which connect with more important veins.

Lixiviation. See Leaching.

Loadstone. Magnetic iron ore.

Loam. Any natural mixture of sand and clay which is neither distinctly sandy nor clayey.

Lock gate. A water-tight door used to hold back water likely to flood mine workings; more especially in deep lead mining.

Lode. A vein containing ores.

Long-horn. The direction of a breast in coal mining in relation to the cleats: about half way between face-on and half-on.

Long-tom. A wooden sluice box about 24 ft long, 2 ft wide, and 1 ft high, for washing auriferous gravel.

Long ton. 2240 lb.

Long-wall. A method of working comparatively thin seams of coal in long faces, by which means all the coal is won with the exception of shaft pillars and main road pillars.

Low grade. Not rich in valuable minerals.

Lump coal. All coal passing over the main screen.

Lydian stone or Touch stone. A hard black silicious rock used for testing the quality of gold alloys.

Made ground. Ground that has been turned over artificially, mostly for levelling purposes.

Magmatic segregation. The separation and collection of certain minerals from other constituents of a molten rock mass.

Main and tail rope haulage. A system of haulage whereby a set of skips connects two ropes, one known as the main which hauls the full skips out, the other as the tail rope which draws the empties into the mine.

Make of ore. A body of ore in a vein.

Man engine. An apparatus consisting of one or two reciprocating rods to which suitable platforms and grips are attached, used for lowering and raising men in shafts.

Man way. A passage for men to travel along; generally alongside a shaft or winze.

Marl. Clay containing calcareous matter.

Marsh gas. See Fire-damp.

Massive rocks. Igneous rocks.

Match. A train of gunpowder, placed in a straw or paper tube several inches long for igniting explosives: or the touch end of a squib.

Matte. The product obtained when smelting certain kinds of ore, whereby the valuable metals are concentrated as sulphides.

Matte-fall. The amount of matte formed in smelting.

Meridian. There are two kinds: (1) The true, and (2) the magnetic. The latter is variable.

Mesozoic. The second great division of sedimentary rocks.

Metal. Metals are solid at ordinary temperatures, with the exception of quicksilver; are opaque, except in the thinnest possible films; have a metallic lustre; and are better conductors of heat and electricity, and as a rule have a higher sp. gr. than non-metals.

Metamorphic. A change of form from the original constituents of a rock.

Metasomatic. A change in the composition of a rock through chemical means by which a mineral is partly or completely altered.

Mill. Works for crushing, concentrating and amalgamating ores.

Milling. Quarrying ore on the surface in the form of a funnel and sending the broken material down a pass to a drift below for conveyance to a shaft or to the surface.

Mine. Any excavation made for the extraction of minerals, either on the surface or underground.

Mineral. A natural, homogeneous, inorganic substance. Though coal does not come within this definition, it is generally included, since it occurs and is mined as a mineral.

Miner's dial. An instrument for measuring horizontal and vertical angles in mining operations.

Miner's inch. A measure of water varying in different districts, being the quantity of water that passes through a slit an inch high, of a certain width, and under a given head.

Miner's right. An annual permit from the Government to occupy and work mineral land.

Moil. A short steel tool with a diamond point for driving into clefts when levering off rocks.

Molecule. The smallest quantity of an element that is capable of entering into chemical combination.

Monkey. The ram of a pile driver.

Mortar. A vessel in which ore is pulverized.

Mother lode. The main lode of any district.

Mother of coal or Dant. A fine film of carbonaceous matter found in the joints of coal.

Muffle. A thin clay oven heated from the outside.

Muller. The upper grinding iron of a pan.

Mullock. The worthless mineral broken in a mine.

Mundic. Iron pyrites.

Naked light. Any unprotected light. In collieries, any form of light which is not a safety lamp.

Narrow place. A working place in a coal mine which is less than six yards wide and is paid for by the yard of length.

Native metal. Any metal that occurs naturally in that state such as gold, silver or copper.

Natural ventilation. When workings of a mine are so arranged that air currents are produced without resorting to artificial means.

Neck. A cylindrical body of an igneous rock.

Needle. A sharp pointed copper or brass rod with which a small hole is kept open during tamping to permit a train of fine powder to reach the explosive charge.

Nick. To shear the side of a breast after holing.

Nicking. A vertical cutting or shearing up the side of a face of coal so as to free the ends of a block that has been undercut.

Nip. When the roof and floor of a coal seam come close together.

Nitrate. Nitric acid chemically combined with a base.

Nodules. Concretions which are frequently found to enclose organic remains.

Normal solution. In volumetric analysis, one litre contains the hydrogen equivalent of the active reagent weighed in grammes. A deci-normal solution is one-tenth the normal strength.

Nozzle. The nose piece of a blast-pipe for a furnace; or for a water pipe.

Nugget. A natural lump of gold or other metal, applied to any size over two or three pennyweights.

Nuts. Coal less than $\frac{3}{4}$ in. and over $\frac{3}{16}$ in.

Oil shale. Shale containing so much hydrocarbon as to be worth while distilling for oil.

Open cut or open cast. Surface mining.

Ore. Any mineral or mineral mass from which it may be reasonably expected that metals or metallic compounds can be extracted on a commercial scale. The term is sometimes applied to other minerals than ores which are specially mined, e.g. phosphates.

Out-bye. In the direction of the exit of a mine.

Outcrop. That portion of a vein, bed, or rock appearing at the surface.

Output. The product of a mine.

Over weight. In longwall mining of coal when the strata overhead settle down on the gob.

Overburden. The soil or rock overlying a mineral deposit.

Overcast. A passage made for one current of air to pass over another current.

Overhead stope. The ordinary way of mining ore overhead in steps.

Overlap fault. A reverse fault in which a part of the faulted rock doubles over the other faulted end.

Oxide. A chemical combination of oxygen and a base.

Pack wall. A wall of stone built on either side of a mine roadway to support the roof.

Paddock. (1) An excavation made with the object of obtaining wash dirt in a shallow alluvial deposit. (2) A place near the mouth of a shaft where ore is stored.

Palaeozoic. The oldest series of rocks in which fossils of animals occur.

Pan or dish. A sheet iron dish used for panning off gold or tin.

Panel. A large rectangular block of coal 30 to 40 acres in area, worked as a separate mine, and separated from other parts of the colliery by barrier pillars.

Parrot coal. A coal that splits or cracks with a chattering noise when on a fire.

Parting. (1) Any thin interstratified bed of shale. (2) A side track or turn out in a haulage road. (3) To separate silver from gold when assaying.

Pass. A built-up passage in a stope for broken material to pass to the drift below.

Pass-by. A siding where cars pass one another on a single rail track.

Patent fuel. Small coal bonded with 8 to 10 per cent tar and compressed into bricks.

Pay dirt. That portion of an alluvial deposit that contains gold in payable quantities.

Pay ore. Ore which can be mined and treated at a profit under existing circumstances.

Peas. Small coal about ½ to ¾ in. cube.

Peat. The decomposed organic matter formed in bogs.

Peg out. Posts or heaps of stones placed at the corners of a claim.

Penstock or Forbay. A pressure box, the intake for a water-pipe leading to a hydraulic nozzle so as to give a certain head or pressure for hydraulic sluicing.

Penthouse or Pentice. A protection overhead for miners while sinking a shaft.

Percussion table. A table for concentrating fine ore which is given constant jolts.

Peter-out. To decrease gradually in thickness.

Picking table. A stationary or revolving table on which ores are sorted by hand into their several classes.

Pig. A piece of iron or lead cast in a long iron mould.

Pigsty timbering. Hollow pillars built up of logs of woods laid crosswise, for supporting heavy ground.

Pillar. A block of solid ore or coal left (1) along the side of a shaft, (2) between the floor of a drift and the roof of a stope when mining has ceased, (3) as a support for the sides of a lode or the roof of a coal seam, (4) as a barrier between adjacent mines, or parts of the same mine.

Pillaring. The process of extracting pillars of coal.

Pinch. A contraction in a vein.

Pinched out. When a vein runs out to nothing.

Pipe clay. A soft white clay.

Pit. (1) A shaft. (2) The underground portion of a colliery. (3) An excavation for obtaining gravel.

Pitch. The pitch of a shoot of ore is the direction along the strike towards which it inclines.

Placer. An accumulation of valuable minerals in an alluvial deposit on the surface.

Plane table. A simple surveying instrument by which one can plot the survey on the field.

Plant. The machinery, workshops, offices, shafts, etc., in connection with a mine.

Plat. A chamber excavated at the junction of a cross-cut with a shaft.

Plunger. The ram of a force pump.

Poling. Refining molten copper by stirring it with green poles.

Poll pick or miner's pick. The longer end is pointed, while the shorter end is hammer shaped.

Poppet head. A pyramidal headframe for supporting the sheaves for the hoisting ropes, and a brace for landing ore.

Prill. An extra rich piece of ore.

Primary ore. Ore which retains its original form and composition.

Prime. (1) When water passes into a steam cylinder together with the steam. (2) To fill the suction pipe of a pump.

Primer. A cartridge in which the detonator is inserted.

Prop. A vertical stick of timber used as a temporary support for the roof.

Prospect. A mineral property before its value has been determined by exploration.

Prospector. One engaged in searching for minerals.

Pseudomorph. When a mineral occurs in a form belonging to another mineral.

Pudding stone. Conglomerate.

Puddle. To stir gravel mixed with clay in water to get rid of the clay.

Puddling machine. A circular machine for washing pay dirt.

Pug mill. A mill for preparing clay for brick-making, pottery, etc.

Pump tree. Cast-iron pipes, generally 9 ft long, of which the column or rising main of a pumping system is made.

Pyrometer. An instrument for measuring high degrees of heat.

Quarry. A term applied to open cuts for obtaining valuable rocks, mostly employed for constructional purposes.

Quaternary. The post-tertiary period.

Quicksilver. Another name for mercury.

Rabbling. Stirring a charge of ore in a reverberatory furnace with specially designed iron rods.

Race. A channel for conducting water to or from the place where it performs work.

Raff. The coarse ore after crushing by Cornish rolls.

Raff-wheel. A revolving wheel with inside buckets for elevating the raff or tailings.

Rake of skips. A number of skips connected together that form a set or train.

Ram. The plunger of a pump.

Rapper or Knocker. A lever with a hammer at one end and a signal wire at the other. When the wire is pulled, the hammer strikes a loose iron plate.

Raw ore. Ore that has not been calcined or roasted.

Reducing. Getting rid of the bulk of sand when sluicing tin in a sluice box, commencing at the top and working down stream.

Reef. (1) A vein of quartz. (2) The bed rock of alluvial deposits.

Reef drive. In alluvial mining, when a drift is driven in the country rock or reef.

Refractory. Applied to ores not easily treated by ordinary methods.

Refuge hole. A small chamber cut in the side of a haulage way underground for a man to take refuge in while cars pass.

Regulator. A door in a mine, the opening and shutting of which regulates the ventilation to a district.

Regulus. See Matte.

Reserves. Mineral already opened up by shafts, winzes, and drifts which can be broken at short notice.

Resue. To extract a strip of clay or rock on the wall of a vein, so as to have a free face to "fire to."

Retort. A closed iron vessel with a long neck used for distilling quicksilver from amalgam.

Return air. Air that has passed through the workings on its way to the surface.

Reverberatory. A furnace in which the flame from the grate is made to beat down on the charge on the hearth.

Rib. A narrow pillar of coal left as a support.

Riddle. A movable frame having parallel bars against which rock is thrown to separate coarse from fine.

Rider. (1) A guide frame for steadying a sinking bucket. (2) A sliding weight on the beam of a balance.

Riffle or ripple. Obstructions placed on the bottom of a sluice box, or inclined table, to retain the heavier particles of ore.

Right shore. The right-hand side when descending a river.

Rill. The coarse ore at the bottom of a heap.

Rise. Working to the rise is when one works on an up-grade.

Rising main. The pipes conveying water from a mine pump to a higher level or the surface.

Roasting ore. Heating ore sufficiently to cause a chemical change but not sufficiently to melt it; such as chloridizing and oxidizing roasting.

Rob. To cut away or reduce the size of coal or ore pillars.

Rock. A natural mass of inorganic matter composed of one or more minerals, whether loose or consolidated.

Rock breaker. A machine for reducing ore in size by crushing it between powerful jaws.

Rocker. See Cradle.

Roof. The rock immediately above a coal seam that is being worked.

Round coal. Coal in large lumps after the smalls have been separated.

Run-of-mine. Ore as it is mined without sorting.

Rusty gold. Native gold coated with iron oxide.

Saddle reef. A dome-shaped reef, a cross-section of which forms an anticline.

Safety catch. An appliance attached to a cage for holding it up in case the rope breaks or is detached.

Safety fuse. A cord with a core of slow burning powder used for exploding charges.

Safety lamp. A lamp used by coal miners, which is protected from explosive gases by wire gauze and glass.

Salt. A salt is produced by the interaction of an acid and a base.

Salting. Fraudulently enriching mineral matter with the object of deceiving others as to its real value.

Sand lock. A wooden door made in two horizontal sections to hold back slurry if tapped in a deep alluvial mine.

Sand pump. A cylinder with a valve at the bottom attached to a rope, for removing pulverized rock from a bore hole.

Schist. A metamorphic rock having a slaty structure.

Scorifier. A small clay dish used in fire assaying.

Screen. A mechanical apparatus for sizing mineral.

Sealing a mine. The air-tight closing of the entrance to a coal mine resorted to in time of fire.

Seam of coal. A bed of coal.

Second working. In coal mining, extracting pillars after the first working.

Secondary minerals. Those which result from the alteration of primary minerals; mostly found in the zone of secondary enrichment.

Seconds. Second-class ore which requires dressing to bring it up to the required standard.

Sedimentary rocks. Those rocks formed by wind or water.

Self-acting inclined plane. An inclined plane on which full cars haul the empties up hill.

Selvage. A band of clay sometimes found on the walls of veins.

Serpentine. An alteration of certain igneous rocks, such as peridotite.

Set of timber. A full set of timber for a drift consists of a sill, two legs, and a cap. That of a shaft, two wall plates, two end-pieces, and dividers depending on the number of compartments required.

Set-off. An iron projection from the main pump rod of a Cornish pump to which the bucket pump rod is attached.

Shaft or pit. A vertical or inclined hole in the ground used for gaining access to underground workings.

Shaking table. An inclined table for concentrating fine ore which is given short, quick shakes.

Shale. A fine grained rock with fissile structure.

Shear zone. A belt of country along which rocks have been crushed but not clearly displaced as by a fault.

Shearing. See Nicking.

Sheave. A wheel with a groove round its circumference as a guide for a rope.

Shift. The number of hours worked continuously.

Shoad stones or Floaters. Loose pieces of the outcrop of a deposit that have been shed.

Shoes. (1) Steel guides fixed to the sides or ends of a cage or skip which run on skids. (2) The upper working face of a stamp or grinding mill.

Shoot of ore. A run of rich ore in a vein.

Shooter or Shot firer. The man who fires a charged hole after satisfying himself that the place is free from fire-damp.

Shooting in the solid. Firing shots when the only free face is that at the breast.

Short-horn. When the breast of coal is worked half way between end-on and half-on.

Shortwall machine. A coal cutter for use in bords which when once the cutting part has made the sumping cut is drawn across the face automatically by ropes, undercutting as it proceeds.

Sickening. A coating of impurities on quicksilver that retards amalgamation or the coalescence of globules of quicksilver.

Sill. A log placed across the bottom of a drift for the legs of a set to stand on.

Sinker bar. The iron rods used in rope boring which are placed between the rope socket and the jars.

Siphon. A bent pipe, one limb of which is longer than the other. Used for drawing water over a height of about 26 feet.

Skewback. The inclined seating from which an arch springs.

Skid. Wooden or steel guides for cages or skips to run on.

Skip. An elongated vessel in which ore or water is raised from a shaft.

Skirting. A road driven next to a fall of stone or an old fallen-in place.

Sky-shaft. The continuation of shaft timbering above the surface.

Slabs. Split pieces of timber, from 2 to 3 in. thick, 6 to 9 in. wide, and 4 to 6 ft long, which are placed behind sets of timber in drifts and shafts.

Slack. Small coal that passes through a ¾ in. screen.

Slag. The molten refuse from smelting operations.

Slant. An inclined roadway.

Slate. A compact clay which has assumed a regular laminated structure.

Slickenside. The polished striated surface on opposite walls of a joint sometimes found in veins.

Sliding wind-bore. The bottom or suction pipe of a sinking set which can be lengthened without altering the whole column of pipes.

Slime or Sludge. The fine pulp from a mill or bore-hole.

Sluice. A channel cut in the bed-rock along which the gravel passes from the face to the sluice box when hydraulic sluicing.

Sluice box. A trough containing riffles for catching gold, or tin.

Sluice head. A measure to gauge the quantity of water that flows in a channel.

Snag. A log wholly or partly buried in alluvial.

Snore-piece. See Wind-bore.

Sollar. A platform in a shaft for a ladder to rest on.

Sounding. Knocking on the roof with a hammer to hear if it is drummy and unsafe.

Sow. A deposit of iron at the bottom of a blast-furnace.

Spall. To break up rocks with a large hammer.

Spanish windlass. A loose log to which a rope to be tightened is held to the ground by two inclined poles. Handspikes are inserted in holes at the ends of the log, by means of which it can be turned, and when the rope is sufficiently tight, a stick is placed horizontally between the handspikes and inclined poles to keep the log from reversing.

Spar. A name given to certain crystalline minerals such as calcspar, feldspar, and fluorspar.

Sparge pipe. A water pipe with small holes in it for spraying minerals, as in a trommel.

Spear wedges. Long wooden wedges used for centring iron tubbing, and which help to pack up the space between the tubbing and the rock.

Special places. Development and difficult places where coal cannot be won so easily as in ordinary working places.

Specific gravity. A comparative degree of weight: that of water being taken as unity.

Specimen. A picked piece of mineral.

Speiss. An artificial combination of arsenic or antimony with iron, copper, nickel, etc.

Spelter. The commercial name for zinc.

Spent shot. A charge hole that has been fired, but did not do the work it was intended to do.

Spew over. An extension of mineral matter on the surface past the ordinary limits of a lode.

Spider. An iron candlestick used by miners that can be stuck into wood or hung up on rock.

Spilling or forepoling. A process of timbering through loose ground.

Splint. An inferior dull looking laminated coal, intermediate between cannel and bituminous coal.

Split. To divide an air current into two or more separate currents.

Spoil. Debris from a coal mine.

Spoon. A slender iron rod with a disc shaped projection set at right-angles to the rod; used for scraping drillings out of bore holes.

Sprag. A short wooden prop set at an angle to support coal during the process of holing.

Spring of an arch. The rise or vertical height between the skewback and the highest part of the arch.

Springing. See Chambering.

Spud. (1) Wooden or iron legs situated at the stern of a bucket dredge to give support when digging, and also to enable the dredge to move forward when desired. (2) A horse-shoe nail with a hole in the head, which is driven into mine timber or a wooden plug fitted into the roof to mark a surveying station.

Square set. A method of timbering wide cavities in which the members of a set are at right-angles to one another, forming a hollow cube.

Squeeze or Creep. The gradual coming together of the roof and floor in a mine, due to the weighing action of the roof or a tender floor.

Stack. A high chimney built on a separate foundation.

Staging. A temporary platform on which to work.

Stalactite. An icicle shaped appendage of mineral matter suspended from overhead.

Stalagmite. Lumps of mineral matter which accumulate on the ground beneath a stalactite.

Stall. A working place in longwall coal mining.

Stall gate or road. Branches off cross-headings to each stall.

Stamp. A large pestle raised mechanically: used for pulverizing ore in a mortar.

Station. See Plat.

Steam coal. A hard free burning, non-caking coal, that burns with a white ash.

Stemmer. The wooden or copper bar used for ramming down tamping.

Stenton. A connection made between parallel roadways for ventilation purposes.

Stratum. (Plural: strata) A layer or bed of rock.

Stock. Rock impregnated with ore.

Stockwork. A network of small veins none of which can be mined separately.

Stone coal. Anthracite.

Stook. A block of coal a few yards square left to support the roof in certain stages of pillar work.

Stoop. A pillar of coal.

Stope. Working a lode in the form of steps.

Stopping. An air-tight wall built across a mine passage in order to direct the air current.

Streaming-down box. A small sluice box for cleaning alluvial gold or tin.

Strike of a lode or bed. The course of a lode or stratum in respect to the meridian.

Strip. To remove the overlying strata of a bed or vein.

Studdle. A piece of timber placed vertically between two sets of timber in a shaft.

Stull. A worked out stope.

Stull piece. Timber used in stulls to support the walls.

Stump. The pillar left between the roadway and each bord turned off from it.

Sulphate. Sulphuric acid combined with a base.

Sulphide. A combination of sulphur with a base without the addition of hydrogen.

Sulphuret. An old name for sulphide.

Sump. The lowest part of a shaft into which water drains.

Swabstick. A short wooden rod bruised into a sort of brush at one end, used for cleaning out drill holes.

Sweep rods. The rods that connect the balance bob of a Cornish pump with a horizontal engine.

Sweet roast. To roast dead or completely.

Swinging supports. Supports for the sweep rods, when the distance between the balance bob and engine is great, which swing backward and forward.

Syncline. Trough-like. The opposite to an anticline.

Tail race. The ditch through which the tailings from hydraulic sluicing are carried away.

Tail rope. A rope working in conjunction with a main rope in a system of underground haulage; or as a balance in a shaft.

Tailings. The detritus from sluicing or ore dressing works.

Tamping. The material used for filling a charged hole.

Tamping bar. A copper or wooden rod for ramming down tamping.

Temper. To change the hardness of metals by first heating and then plunging into water, oil, etc.

Temper screw. A long screw used in rope boring by means of which the tools can be lowered gradually as the hole deepens.

Tertiary. The third great division of rocks in which the highest class of vertebrate animals first appear.

Test. An iron framework which is filled with bone ash for cupelling on a large scale.

Test tube. A thin glass tube closed at one end for testing substances in the wet way.

Theodolite. An instrument used in surveying for taking both vertical and horizontal angles.

Throw. The vertical motion of a fault.

Tip. A place where ore or mullock is dumped.

Tippler. A platform with rails, attached to an axle, on to which a car is run which can be titled over and emptied.

Token. A ticket attached to a tub of coal to show who had mined it.

Tommy dod. A series of small pulleys with vertical axles placed between the rails at a curve so as to keep an endless rope in place.

Top coal. The upper portion of a thick seam, often left to be extracted at the same time that the pillars are drawn.

Tramp steel. Pieces of steel such as tools that get mixed up with ore.

Trap rock. Igneous rocks that occur in the form of steps.

Travelling weight. See Underweight.

Trepan. A compound bit used in the Kind-Chaudron method of shaft sinking.

Tribute. A method of working mines on contract whereby the miners receive a certain proportion of the value of the products won.

Triple entry. A system of opening a coal mine by driving three parallel main entries.

Trommel or Drum. A revolving perforated sheet metal cylinder or cone used for washing or screening ore.

Tromp or Water blast. A means of ventilating small mines by causing water to fall down a wooden pipe and drag in air with it.

Trough fault. A mass of rock let down between two faults.

Truck system. Paying men by food instead of with money.

Tub. A car or truck used in coal mining.

Tubing. The lining of bore holes with wrought-iron tubes to prevent the sides from falling in.

Tubbing. Cast-iron sections for lining wet shafts.

Tuff. The finer kinds of volcanic ash, generally found in more or less horizontal layers.

Tunnel. Strictly speaking, a horizontal passage which comes to the surface at both ends: but in mining parlance a passage that may have but one end open to the surface.

Turbine. A rapidly revolving wheel impelled by water or steam impinging on curved blades.

Turn-out. A siding or branch for tubs or trucks on a haulage road.

Turntable. A horizontal revolving table used for turning cars in different directions.

Tut work. Breaking ground at so much per foot or fathom.

Tuyère. The nozzles through which air passes into a furnace.

Tye. An inclined table for dressing ores.

Uncomformability. When one set of rocks does not lie at the same angle as those on which it rests.

Undercast. A passage made for one current of air to pass under another.

Undercut. To hole coal.

Underhand stoping. Working out ground underfoot in steps.

Underlie. The slope at which a vein lies at right-angles to its strike.

Underlie shaft. An inclined shaft more or less parallel to the vein it is intended to serve.

Underweight. In longwall mining, when the roof rolls forward and presses on the rib of coal that has been undercut.

Unit. The unit of a metal is 1 per cent of whatever ton is used. If a long ton of 2240 lb the unit would be 22·4 lb: but if a short ton of 2000 lb, the unit would be 20 lb.

Upcast. A shaft through which the foul air of a mine reaches the surface.

Vat. A large tub of wood, iron, or concrete used for leaching or precipitation.

V-bob. A two-limbed crank for connecting pump-rods when the angle of the shaft is changed.

Vein. A sheet of mineral matter which has been formed since the rock in which it occurs.

Vein-stone. The non-metallic portion associated with ores in a vein.

Vernier-scale. One scale that moves alongside another with a slightly different graduation to enable one to read intermediate distances.

Viewer. The general manager or mining engineer of one or more collieries.

Volley firing. When several shots are fired at the same time.

Walking or working beam. A beam that works up and down on a vertical support, one end of which is connected with the motive power and the other with the pump rod.

Wall plates. The timbers on the longer sides of a rectangular shaft set.

Wall within walls. When a vein has three or more walls due to the reopening of the original vein.

Walling stage. A movable stage from which a shaft can be lined with bricks or tubbing.

Wash dirt. That portion of alluvial in which most of the gold is found.

Water hammer. A hammering noise caused by the intermittent escape of gas through water in pipes.

Water level. The level at which water is met when sinking. (Not to be confused with sea level.)

Water ring. A gutter cut in the wall of a shaft to collect water dripping down from which it is led by a pipe to a suitable place.

Water wheel. A wheel provided with buckets which is set in motion by the weight or impact of a stream of water. They may be overshot, undershot, or breast wheels.

Weather. When a rock decomposes and crumbles on exposure to atmospheric agencies.

Wedging curb. A ring of wood or cast iron in sections, wedged tightly in place to form a water-tight joint upon which tubbing is built.

Weir. A dam over which water flows.

Well. The well of a furnace is the hollow at the bottom in which metal collects.

Welsh-bord. A bord in which the waste is placed in the middle and a roadway is kept open on either side.

Whim. A large horizontal drum supported by a suitable framework, round which the hoisting rope of a shaft is wound. Generally worked by a horse.

Whip. A post fixed in the ground at an inclination on the top of which is a sheave, the throat of which is over the centre of a hoisting compartment. The rope passes over the sheave, under a holding-down sheave, and is attached to a horse which pulls the full bucket up and eases the empty bucket down

White-damp. Mostly carbon monoxide and nitrogen.

White tin. The commercial name for metallic tin.

Whole working. Working in the whole, or first working of a seam by which it is divided up into pillars.

Willoughby. A sort of jig in which alluvial tin is concentrated for the market.

Wind-bore. A strainer at the bottom of the suction pipe of a pump.

Windlass. A horizontal barrel with a crank handle at one or both ends; used for winding a rope in a shallow shaft or winze.

Windsail. The top part of a canvas pipe used for conveying air down a shalllow shaft.

Windy. Applied to a rock, generally a shale which frets away when exposed to the air.

Windy shot. When a portion of the gases from a blast escapes through the tamping or crevices in the rock, without doing the required work.

Winning-off. A leading heading or drive in advance to win room for bords. Any leading drift is termed,a Winning.

Winnowing. Tossing up fine dry auriferous material in the air where there is a draught, and catching the heavier particles which are not blown away.

Winze. A more or less vertical passage sunk from one level to the next below.

Working home. Working towards the entrance of a mine instead of away from it.

Working in the whole. Driving headings and bords in a coal mine.

Wrought iron. Iron containing the smallest amount of carbon.

Zone of secondary enrichment. That portion of a lode where minerals from the denuded upper part have been dissolved and redeposited in another form.

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